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MINING METHODS OF THE HOLDEN MINE, HOWE SOUND CO., CHELAN DIVISION, HOLDEN, WASH.

BY

R. L. SODERBERG

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MINING METHODS OF THE HOLDEN MINE, HOWE SOUND CO., CHELAN DIVISION, HOLDEN, WASH.[1]

By R. L. Soderberg[2]

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2/ Blast-hole engineer, Howe Sound Co. - Chelan Division, and one of the consulting engineers, Bureau of Mines.

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INTRODUCTION

This paper is one of a series on mining methods and costs being published by the Bureau of Mines. It describes briefly the mining methods employed since the start of operations in 1937, with emphasis on the diamond-drill blast-hole method of ore breaking and slusher-drift method of ore drawing at the Holden mine of the Howe Sound Co., Chelan Division, Holden, Wash.

The Holden mine is in the north-central part of the Cascade Range, 11 miles up Railroad Creek from Lake Chelan. The nearest railroad is at Chelan Falls, 40 miles by boat and 4 miles by highway from the mine. All freight comes into the mine and all concentrates are shipped out via Chelan Falls. The altitude of the main haulage level is 3,500 feet.

Railroad Creek Valley is covered with a dense growth of fir and spruce, while most of the surrounding mountains are rocky, with sparse timber growth.

ACKNOWLEDGMENT

The writer wishes to acknowledge the cooperation and assistance of the executives and members of the staff of the Howe Sound Co., Chelan Division, in the preparation of the manuscript and illustrations of this article.

HISTORY[3/]

Maj. A. B. Rogers, a locating engineer for the Great Northern Railway, while searching for a pass over the Cascades, was the first to note that a mineralized area existed where the Holden mine now operates. In 1887, he came up Lake Chelan to Railroad Creek and ran a grade line up the valley until he came to the Crown Point Falls, a few miles from the crest of the Cascades, but found the cliffs too steep for a suitable grade.

When he returned to Seattle in the fall, Major Rogers told his friends, the Denny family, about the large, red oxidized zones he had seen on both

3/ Curzon, John J., (manager, Howe Sound Co., Holden, Wash.), Changing Mining Methods at the Holden Mine: Trans. Am. Inst. Min. and Met. Eng., vol. 163, 1945, pp. 73-95.

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valley walls along Railroad Creek. They grubstaked J. H. Holden to explore the area and stake any promising ground. Holden staked the first claims in 1892 for himself and the Dennys. A. H. Murdock of Chelan, and a partner, W. P. Robinson, staked adjacent ground. These two groups of claims are a large part of the mining property being worked today.

Four drifts, none of which hit commercial ore, were run along the mineralized zone by the first prospectors. The Alder tunnel, just above creek level at an altitude of about 3,400 feet, was driven for a distance of 466 feet in glacial till. The second one, the Robinson tunnel, is about 50 feet long, driven in silicified sedimentary rocks. The third, consisting of two short drifts, known as the "Gold Bug", at altitudes of 4,744 and 4,770 feet, in Copper Creek Valley, cut silicified rocks containing minor pyrite. The fourth drift, on the west slope of Buckskin Mountain at an altitude of 6,100 feet, was driven 15 feet in a lens of pyrrhotite.

W. S. Denny lost control before 1910, when an important decision was handed down by the Washington State Supreme Court. It held that an alleged grubstake of 11 years' standing was not a sufficient basis to claim an interest in a mining property which had been developed and upon which large sums of money had been expended to put into production, and therefore denied the claims of W. S. Denny, of Seattle, and gave J. H. Holden perfect title.4/

During the period between 1892 and 1928, many attempts were made by well-known mining companies to bring the property into production. In 1928 the Britannia Mining & Smelting Co. optioned the property, and in 1930 a subsidiary, the Chelan Copper Mining Co., was formed for exploration purposes. Several drifts were run in the next 2 years; but it was not until March 1937, when the Chelan Division of the Howe Sound Co. took over, that active development of the mine was started. During that year, a concentrator, docks, tug and barges, 11 miles of road, 54 miles of power line, shop buildings, and living accommodations were built. The mill started operating in April 1938.

GENERAL GEOLOGY

The Holden mine is in a roof pendant of sedimentary rocks that has been subjected to intense metamorphism. The sediments were folded into very tight folds, with accompanying faulting and fracturing across the bedding planes. The average strike of the ore zone is N. 24° W., with a dip of 65° to the SW., parallel to the bedding planes of the sediments. To simplify mining lay-outs, the coordinates have been arranged to show the strike as east-west and the dip south. (Hereafter, all bearings and directions will be referred to in mine coordinates.)

The gossan zone crops out across the north face of Copper Peak. In some places primary sulfides are found on the surface. The rocks are mainly quartzite, gneiss, and schists, cut by many granitic dikes ranging in size from the major dike 50 to 80 feet wide, down to very small ones of a few inches or less. The major dike has a strike of about N. 80° W. and dips vertically, replacing

4/ Northwest Mining News, December 1909.

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the ore body completely for a vertical depth of 100 to 150 feet just below the 800 level. The great magnitude of this dike was not known until just before mining operations were begun at that horizon. The numerous minor granite dikes cutting the ore body makes it more difficult to get fair samples. In mining operations all the dikes within the ore zone are mined.

The minerals found are quartz, biotite, magnetite, galena, sphalerite, chalcopyrite, pyrrhotite, and gold in their order of deposition. Chalcopyrite, sphalerite, and gold are the only commercial minerals. Minor minerals are covellite, chalcocite, malachite, native copper, limonite, and pyrolusite, but they have no economic importance. Gold is distributed throughout the mine following the copper trend roughly. The zinc is generally concentrated along the foot wall, which is well-defined, while the hanging wall is determined by assay.

At the east end of the mine, the ore ends abruptly against what may be a preore dam fault, striking N. 20° W. and dipping east at a steep angle. It is very indistinct; in most places it cannot be found because late granite intrusions have entirely obliterated all evidence of it. Occasional breccia is found, but there is no gouge whatsoever. The sediments and the ore have a definite drag toward the south all along this fault, and the highest-grade ore is found adjacent to it, with a gradual decrease in grade to the west. The ratio of the amount of gold to copper also decreases to the west. The ore zone at the east end of the mine has a high sulfide content and a low silica content, as compared to the west end; consequently, it is softer and less abrasive. This is especially noticeable in drilling the blast holes, as the footage obtained per drilling shift is a little lower and the footage per bit is very much lower in the west end.

The sedimentary rocks east of the dam have some of the characteristics of those that make up the ore zone but contain no ore. Pyrite is disseminated throughout large areas east of the dam and is also found well within the foot wall of the ore zone.

Post-mineral faults are numerous. One main fault and innumerable small ones occur in the west end of the ore zone. The main post-mineral fault is normal, strikes N. 32° E., and dips 62° to the SE. The offset is 260 feet vertical and 90 feet horizontal, with the block to the west moving south. The numerous small faults have the same movement. Most of the west end is so badly cut by faults and granite dikes that it is very difficult to find an ore body large enough to have commercial value.

PHYSICAL CHARACTERISTICS OF ORE AND ENCLOSING ROCKS

The ore body as first known was 900 feet long, 1,100 feet high, and 30 to 80 feet wide, dipped 60° to 70°, and extended from the main haulage level (1500) to just below the 300 level. The ore and the country rock were of nearly uniform hardness and stood well without support. The grade was rather low, indicating that fairly large daily tonnages would be necessary to make the operation profitable.

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With these points in mind, the powder-blast method of ore breaking, combined with the bulldoze chamber method of ore drawing, were chosen as being the most suitable. This method promised high tonnage per man-shift and low cost per ton of ore mined. The width of the ore within this block of ground averaged about 60 feet, which was ideal for powder blasting. Widths of less than 35 feet are too narrow for this system. The ore was very hard and dense, with very little fracturing or jointing to help the fragmentation to any marked degree, but the tremendous charges of dynamite were expected to crush the ore to a size that could be handled by the bulldoze chambers.

The grizzly openings were 30 by 60 inches on the upper levels where the rock fell 550 feet down a raise to control chutes, and below this, fell about 400 feet to a chute on the main haulage. As the mining operation approached the main haulage, the grizzly openings were reduced to 24 by 60 inches to offset the shorter fall.

METHODS OF PROSPECTING AND EXPLORATION

Exploration underground, a combination of drifting and diamond drilling, was carried on along the strike and down the dip of the known ore body.

On the upper levels, the ore zone was entered from the west end; drifting was carried east along the strike. When the known ore zone ended, the exploration drifts were driven east, with regularly spaced diamond-drill holes both north and south. Little, if anything, was known about the dam fault along the east end at that time, and on the 700 level the exploration drift was driven many hundreds of feet east in an attempt to find more ore. Again on the 1100 and 1500 levels, exploration and diamond drilling were carried on east of the fault; but by this time, its presence was apparent, as the drag in the ore had been noticed, and small, granite-healed breccia had been found. There were sedimentary rocks to the east, but they carried no commercial ore and, on the 1500 level, were abnormally high in ferromangnesian minerals.

Before development was started below the main haulage level, exploration drifts were driven in the hanging wall parallel to the ore zone and 200 to 350 feet from it. Deep holes were then drilled from the exploration drifts. Large stations were cut every 200 feet along the strike to handle a 20-foot drill rod, and 3 holes were drilled north from each station toward the foot wall at minus 30° to minus 87°. The deepest hole was 875 feet. These holes were all surveyed carefully and all tended to cut normal to the structure of the sediments. When revealed by lower level development, the holes were found to be surprisingly close to their surveyed position.

METHODS OF SAMPLING AND ESTIMATION OF TONNAGE AND VALUES

On the upper levels of the mine before much was known about the dip and strike or about the granitic dikes cutting the ore, the drifts were run in the ore as much as possible. From these main drifts, grab samples were taken from each round before it was mucked and were checked with a chip sample across the face. All crosscuts were sampled with a grab from the muck pile and a chip along either or both ribs, giving an accurate sample of that particular section. All powder drifts and crosscuts were sampled in the same

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way. As many samples as possible were taken from each block of ground to be mined. This greatly facilitated prediction of mill heads.

Diamond-drill core holes, EXE size, were drilled every 2 to 2-1/2 chute sections, 40 to 50 feet, north and south into the foot and hanging walls. The geologist in charge was responsible for seeing that nothing was missed. Occasional holes were drilled an extra 100 feet in length so as not to miss lenses outside the main zone. To check on any questionable area, core holes were drilled from a powder drift or an ore pass. The core was logged carefully and filed. Any mineralized core was split - one-half for a permanent record and one-half for assay. Thus any area in question could be checked by inspecting or assaying the filed core.

When it was found how accurately the foot wall of the ore could be projected, the development drifts were driven 25 to 40 feet within the foot wall of the ore body, and horizontal and plus 45° diamond-drill holes were drilled south through the ore body into the hanging wall at 50-foot intervals. This left quite a blank space between the drilled areas, as the level interval was 175 feet vertically, but the regularity of the ore values made further drilling unnecessary except in special cases.

In laying out the ore outline for any block of ground, the geologist works from a composite plan of all levels within and bordering the block, and from vertical sections every 50 feet normal to the strike and on the line of the drill holes. From the sections, the ore outline is tentatively set with quite regular walls. Each section is calculated separately for total copper and gold units, multiplying the footage by the assay for each sample in every hole within the boundary set. Weighting of holes within any one section is not necessary because of the regularity of the ore body. Each section controls a block 25 feet wide on each side and is given a weight of 1, while a section on either end of the stope, if ending on a section, is given a weight of one-half when the final assay is calculated. The weighted sections are added to get the final figure of total footage and total copper and gold units. The average is obtained by dividing total units by total footage in each case. A correction of 10 percent on all block assays is used to take care of overbreak or sloughing of lower-grade material into the stopes. If the assay obtained is not high enough, it can be corrected by pulling in the hanging wall or cutting out low-grade areas. In outlining the ore to get the proper assays, the mining method must always be kept in mind, for powder blasting, horizontal, or vertical ring drilling have their limitations as to shapes that can be mined. It is easier to follow an irregular ore outline with the vertical ring system than with either of the other two.

The tonnage in a block is calculated from the same plans and sections used in getting the average assay. The sections are planimetered to find their area in square feet, and from this the cubic footage is calculated. As a check, the plans at various elevations are planimetered also, and a separate cubic foot measurement is obtained. A factor of 10.3 cubic feet per ton is used.

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DEVELOPMENT ABOVE MAIN HAULAGE

Haulage Level

The 1500 is the main haulage level, 10 feet by 10 feet in section. It served the area above the 1400 as well as the No. 2 shaft for the lower-level haulage. As the ground was hard and stood very well, little timbering was necessary; but, as mining progressed, the part of the drift paralleling the ore body began taking weight. At present 1,000 to 1,200 feet of drift are timbered.

Shaft

The No. 1 shaft was a vertical shaft 936 feet long. It had two compartments and was used for service only. It was driven simultaneously from the 800 and 1500 levels as a raise, well within the foot wall on the 1500, and at the top it was within 75 feet of the ore body. It was completed early in 1938 and was used until 1945, servicing the 1250, 1100, 800, 700, and 550 levels.

Ore Passes

The main ore passes, 5-1/2 by 11 feet in section, were driven as two-compartment timbered raises and were stripped before being used as ore passes. They were driven from the haulage level to the first grizzly level 900 feet above on 200-foot centers. A main wing and a short wing, 6 by 6 feet in section, were run at plus 52° to plus 55° slope both east and west from the main ore passes and made to intersect the grizzly level at a point 15 feet from the foot wall and on 40-foot centers. The point of intersection was the position for each bulldozing chamber and draw hole to the stope undercut. Thus, one main ore pass served five grizzlies, an effective mining length of 200 feet.

Grizzlies were installed on six separate levels to mine the block of ore - an average of 145 feet vertically for each grizzly level. While the bulldoze chambers and draw holes on the upper levels stood up well and would have drawn more ore, those on the 1400 level stood up hardly long enough to allow the allotted tonnage to be drawn.

Service Raises

Two main service raises supplemented the No. 1 shaft on the lower levels. One was a three-compartment (ore, waste, and manway) incline raise at the west end and served the 1400 and 1250 levels. The other was a two-compartment incline raise at the east end, which served the 1400, 1250, and 1100 levels. These raises speeded the development of the block by supplying more working faces on those levels. Temporary manway and service raises were driven wherever necessary from the main levels to the powder drifts. In many places, ore passes were used for service either before or after having served their intended purposes.

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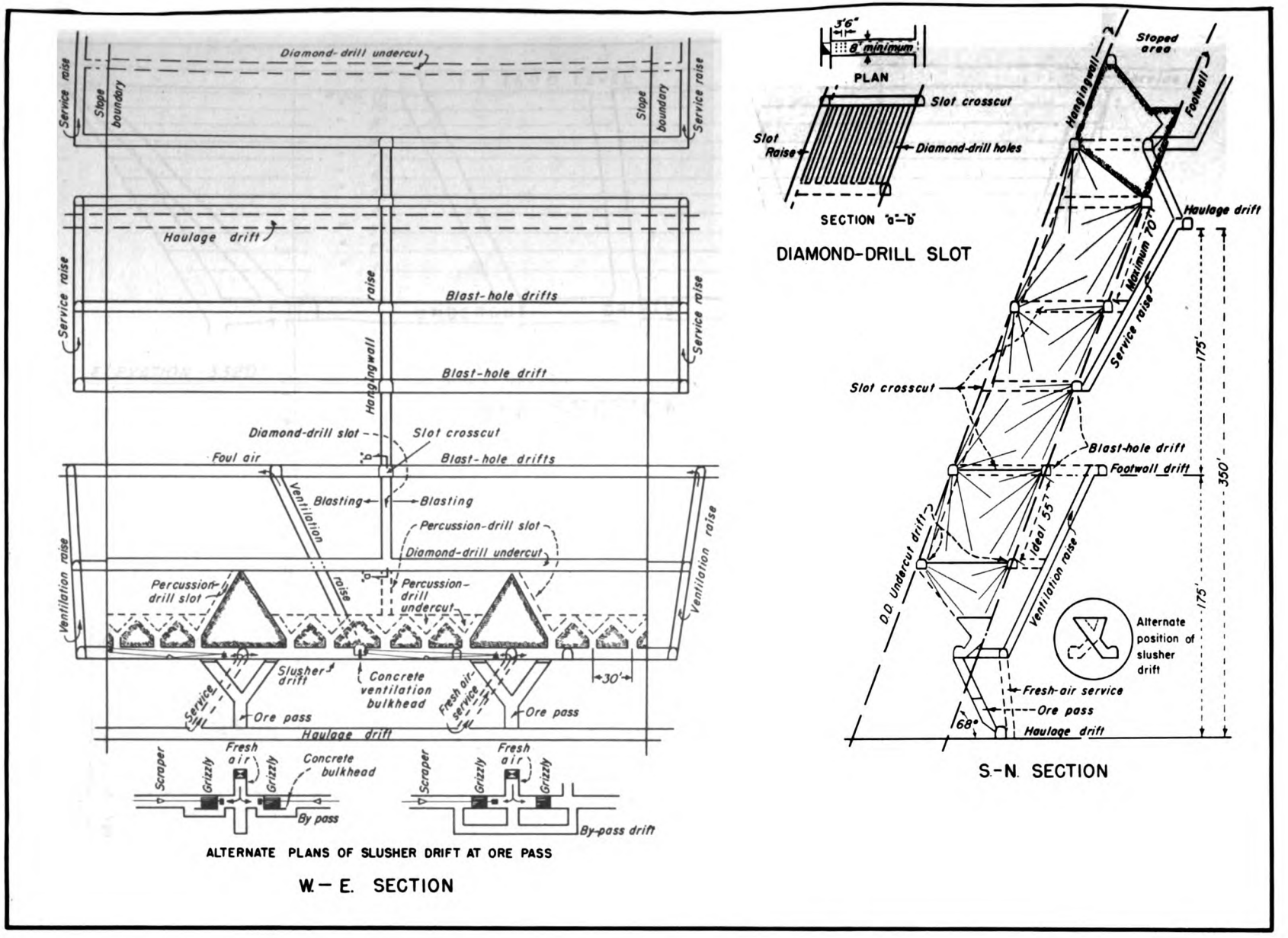

Figure 1. - Ideal section, Noranda-type stope.

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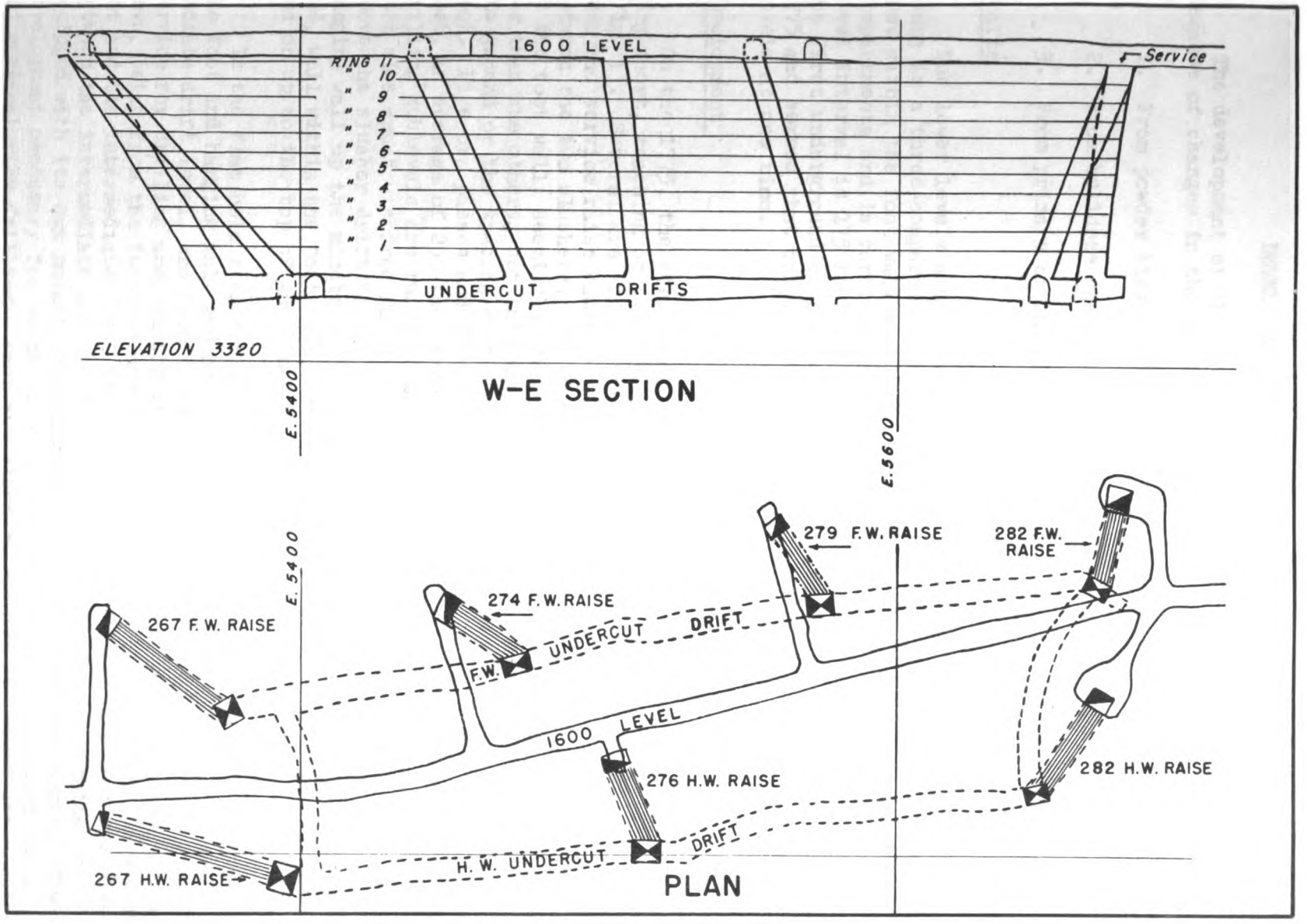

Figure 2. - 280 stope diamond-drill lay-out undercut to intermediate level.

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DEVELOPMENT BELOW MAIN HAULAGE LEVEL

The development of the lower levels differs from that above the 1500 because of changes in the mining methods. They are:

1. From powder blasting to diamond-drill blast-hole mining.

2. From bulldoze chambers to slushers for ore drawing.

3. From primary crushing in the mill to underground crushing.

Shafts

The lower levels are served by the 283 winze and the No. 2 shaft. The winze is a three-compartment shaft, sunk for exploration purposes, and is 75 feet within the foot wall at a minus 65° slope. The No. 2 shaft has five compartments and is vertical and well within the foot wall of the ore. The level interval is 175 feet vertically, with every second one a haulage level. The first underground crusher was installed on the intermediate level below 1775 and served that only. The second installation will serve two haulage levels at one time.

Development

On the 1775, the first haulage level below the 1500, the ore passes are very short, reaching only a vertical distance of 60 feet to the slusher drift (fig. 1). Several are in the form of a Y and serve two slushers back-to-back, with the service raise coming up in the foot wall and a service crosscut going between the two slushers. The slusher drifts were driven both in the ore and in the foot wall; seemingly, those in the foot wall are standing up much better than the others, but this may be because of the more siliceous nature of the ground or the spacing of the draw holes on 30-foot centers instead of 20-foot. Five ore passes and five service raises serve a mining length of 1,270 feet, an everage of 250 feet each. In the Noranda type stopes, 7- by 7-foot drilling sublevels are run on both the foot and hanging walls and 7- by 11-foot slot crosscuts from foot to hanging wall every 50 to 60 feet vertically above the slusher drift to the top of the block. Slot raises are run on the hanging wall up the middle of the stope. Service raises are driven at each end, well within the foot wall, from the undercut to the intermediate level and on up to the top of the block serving each blast hole drift (fig. 1).

In the flat hole ring drilling from raises, the raises are located on the foot and hanging wall and are run from the undercut drift to the intermediate drift level, and from there on up to the next haulage level (fig. 2). Service for drilling and loading in the diamond-drill raises is from the top down, being from the intermediate level down for the area between the undercut and the intermediate level and from the haulage level down for the area between the intermediate and the haulage level. Each drilling raise is equipped with its own movable drilling platform run by an air tugger. The development necessary for ventilation is the same for both the horizontal and vertical ring drilling - each two slushers require one ventilation raise.

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DEVELOPMENT DETAILS

No. 1 Shaft

The No. 1 shaft was designed as a two-compartment shaft, manway, and cage, 8 feet 8 inches by 12 feet 6 inches, outside timber. It is used only for the levels above 1500. As previously stated, it was run as a raise from two different levels - the 1500- and 800-foot. Muck was drawn off through a chute and tramned to the surface at each level. Timber was 10- by 10-inch fir and was carried to within 2 to 4 feet of the face before each blast. Sets were on 6-foot centers with cage compartment lagged on the inside with 4-inch lagging for a muck chute. A control chute was installed between the 1,500- and 1,100-foot levels to shorten the fall of the muck when drawing. Three-inch stopers were used with conventional 1-inch hexagonal steel, starting with a 2-3/8-inch hole and ending with a 1-5/8-inch hole. Powder used was 1-1/8- by 8-inch 40-percent special gelatin. Four miners and a nipper on three shifts could usually set up, drill out a 7-foot, fan-cut round of 37 holes, timber, and blast in 24 hours.

283 Winze

The winze was driven not only for exploration and development, but also for ore hoisting if necessary. It was driven at a minus 65° incline, 75 feet within the foot wall of the ore body, 6 by 17 feet in rock section, 5 feet 4 inches by 14 feet 8 inches, outside timber, with two skipways and a manway (fig. 3). Timber used was 8- by 8-inch fir, with 2-inch lagging on the hanging wall and between the manway and the ore skipway. The posts from the 1775 level down were 6- by 8-inch spruce.

Drilling was done from two bars set up from foot to hanging wall near each rib. The bars were set at an angle of 25° to the horizontal, or normal to the dip. Two C.P. Model 60 hand-cranked machines were mounted on each bar, with an extra on hand in case of a break-down. A 46-hole, V-cut round broke an average of 4.4 feet per round for 140 rounds. The rock was very hard, being a diorite gneiss cut by occasional quartz veins and granite dikes. Conventional 1-inch hexagonal lug steel was used, starting with a casing starter 2-1/2 inches in diameter. After casing was inserted, a 1-3/4-inch bit was used, with 1/16-inch changes ending with 1-3/8-inch steel bit.

Five men were on the bottom at all times. In drilling, one man was always on hand as a trouble shooter to help anyone in need of it. Nine hours were required to drill out a round, 12 hours to muck it, and 5 hours to install a set of timber complete with rails.

The powder that gave best results was a 40-percent gelatin, using 23.33 pounds per foot of advance, or 5.67 pounds per cubic yard in the solid.

Ventilation was supplied by a 15-hp. blower. This delivered air through 16-inch galvanized pipe. Smoke delay was 15 to 30 minutes per round. Water was a very minor problem. The flow was only 20 to 30 gallons per minute for the first 310 feet, then increased to 80 and 100 gallons per minute. Pumps used were 25-hp., rated at 200 gallons per minute at 300-foot head. The

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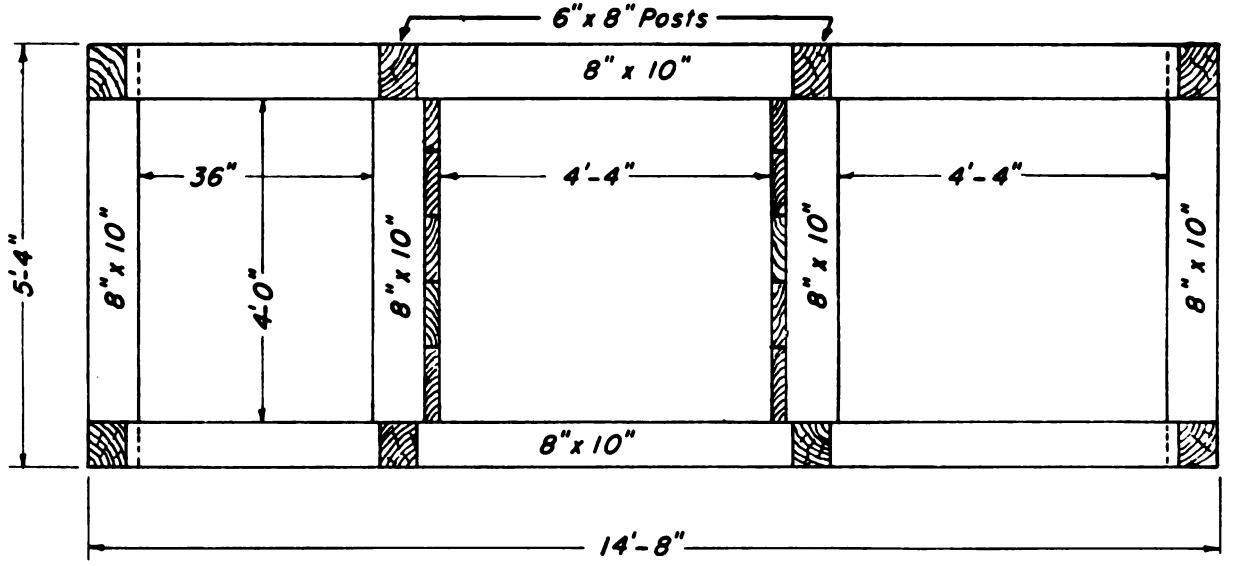

Figure 3. - 283 winze timber.

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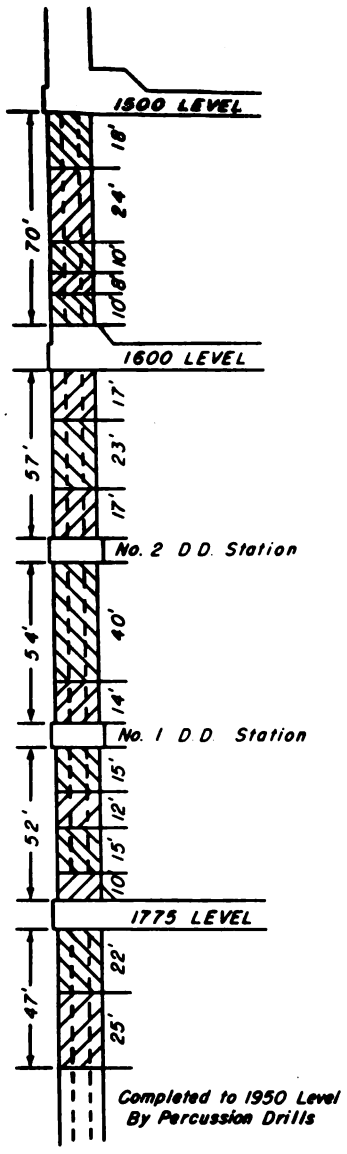

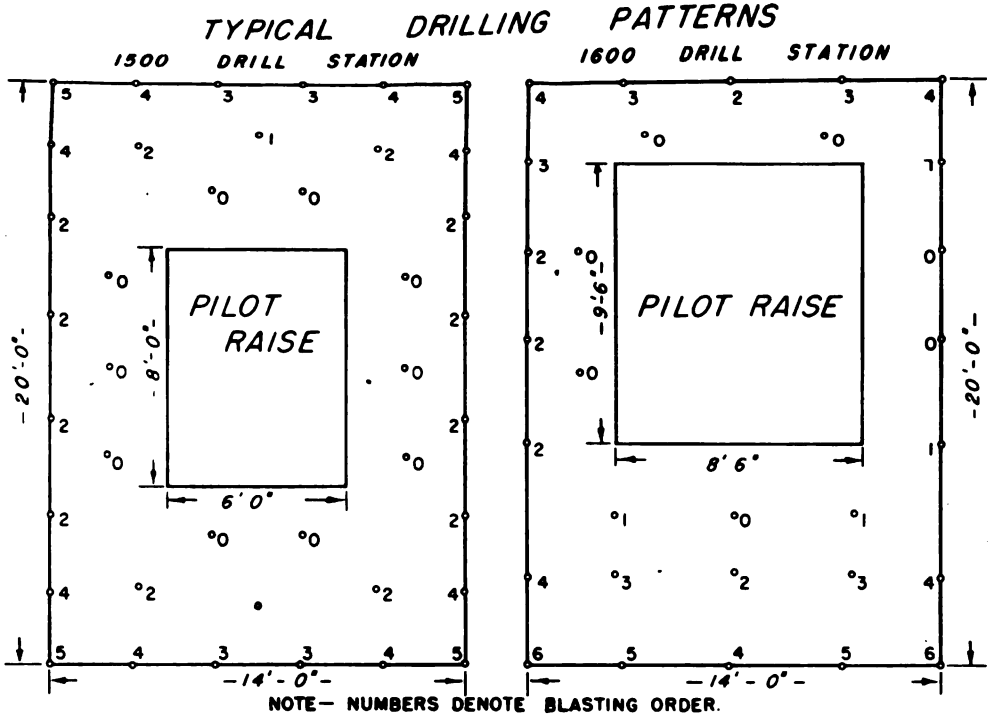

NOTE— NUMBERS DENOTE BLASTING ORDER.

SUMMARY

STATION	NO. HOLE	TONS BROKEN	FT. HOLE DRILLED	TONS PER FOOT HOLE	POUNDS POWDER	POUNDS POWDER PER TON
1500 DRILL STATION	40	1577	2717	0.58	2013	1.28
1600 " "	30	1107	1558	.71	628	.57
No. 2 " "	30	1216	1521	.80	694	.57
No. 1 " "	31	1010	1541	.66	607	.60
1775 " "	30	972	1395	.70	710	.73
TOTALS and AVERAGES	161	5882	8732 *	.69	4652	.79

* ALL HOLES WERE 1-3/16" EXCEPT 2237 FEET OF 1-7/16" HOLE, DRILLED FROM THE 1500 DRILL STATION.

Figure 4. - No. 2 shaft, typical diamond-drilling pattern.

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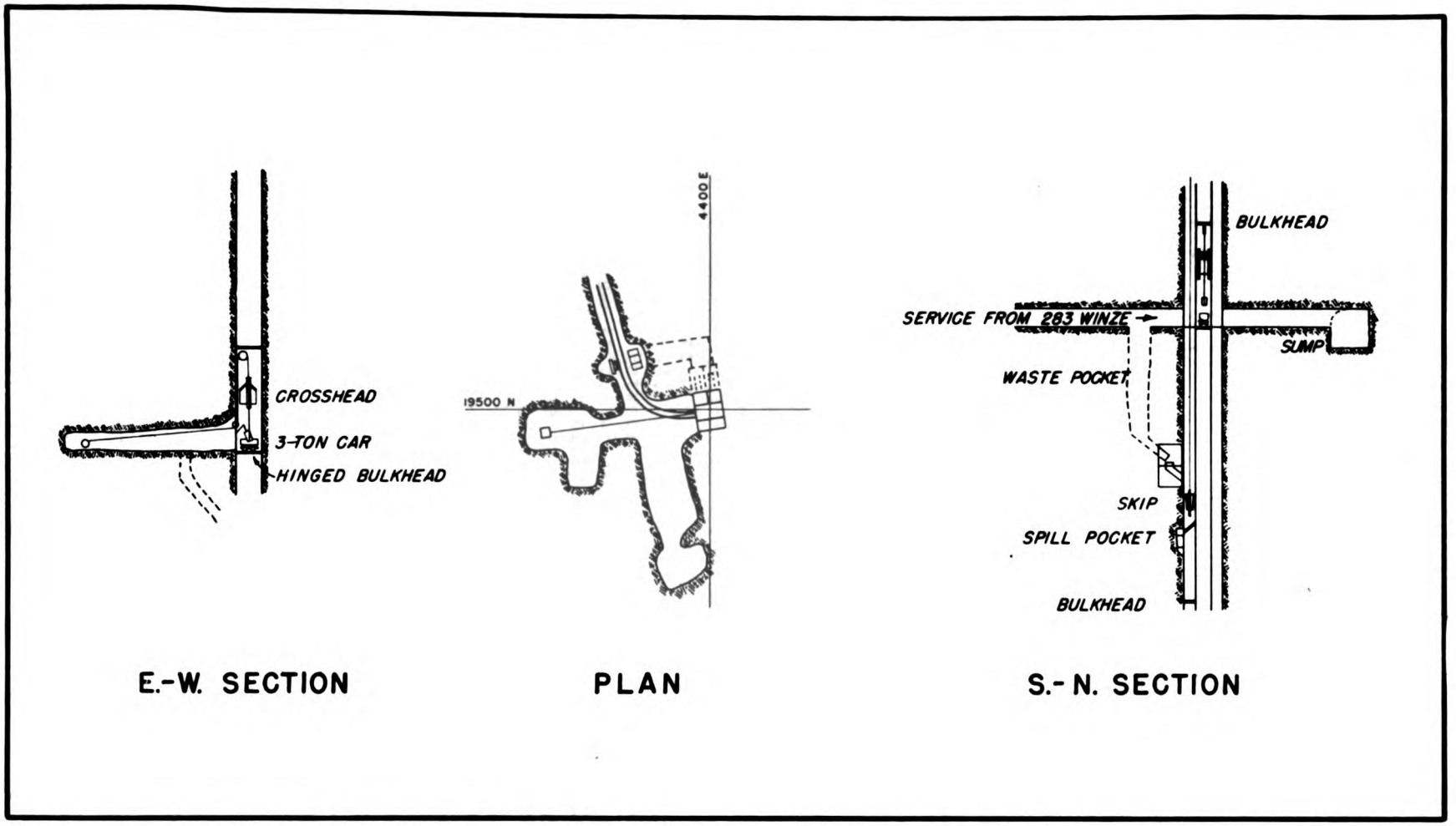

Figure 5. - Method of handling waste rock from shaft-sinking operations.

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sinking hoist was a 40-hp. single drum, with a rope speed of 370 feet per minute and a rope pull of 3,550 pounds with half a drum of cable. It would spool 1,350 feet of 5/8-inch cable. The time consumed, in man-hours, and the advance per round, are tabulated below:

	Drilling	Timbering	Mucking	Track	Air water pipe	Lost time	Total
Hours	1,537	582	2,138	63	39	243	4,602
Percent of total	33.4	12.6	46.5	1.4	0.8	5.3	100
Hrs. per round, average	11.0	4.2	15.3	.4	.3	1.7	32.9
Man-hours	7,433	2,766	10,339	318	195	1,188	22,239
Man-hr. per round	53.1	19.7	73.8	2.3	1.4	8.5	158.8
Man-hr. per foot	12.1	4.5	16.7	.5	.3	1.9	36.0
Total footage							617.7
Feet per round							4.41

No. 2 Shaft (Raising)

The No. 2 shaft has five compartments; is 12 feet 6 inches by 18 feet 10 inches, outside timber; and serves the area below the 1500 level. A two-compartment pilot raise, 6 by 8 feet in section, was driven from the 1600 to the 1500 level to supply a breaking face for diamond-drill holes to slab out the full size of the shaft. The raise was stripped, and diamond-drill holes were drilled from the top down, the longest holes being 70 feet. These holes were loaded from the top, the bottom 8 to 30 feet was loaded on all the holes, which were shot as a round (fig. 4). The broken rock was mucked from the 1600 level and trammed to the winze. Between the 1600 and 1775 levels, where the distance was too great for one lift, two intermediate stations were cut from the pilot raises, reducing the length of holes drilled to about 55 feet. The blasting procedure was the same as the first. Below the 1775 level, a little different scheme was tried. Instead of cutting the stations from the pilot raise, the idea was to drill out and blast the first 60 feet below the 1950, then timber down to the bottom, then cut a little clearance around all four sides of the shaft for the drills, and drill another 60 feet and so on down to the bottom. The first 60-foot lift went well, with only about 3 feet of bootleg around the outside; but the holes along one rib did wander a little, making it necessary to slab a few feet off of the south rib the last 20 feet as the timber was carried down. Instead of drilling out the next two lifts with diamond drills as planned, percussion drills were used just ahead of the timbering.

No. 2 Shaft (Sinking)

The sinking operation as carried on now is conducted so as to be completely free of the operating section above. All service comes to the 2,125-foot level by way of 283 winze. Mucking and service down from the 2,125-foot level are through the center compartment, which is bulkheaded off 80 feet above the level. The muck is hoisted to the 2125, dumped into a 3-ton car, trammed 40 feet, dumped into a pocket; and hoisted by the regular ore skips from the operating part of the shaft (fig. 5).

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A crew consists of seven men; five miners on the bottom, a topman, and a hoistman. A bench round of 40 holes is used, half of the shaft being drilled out and blasted each round. This leaves the muck piled high on the opposite side of the shaft, so that a big percentage of the muck can be scraped directly into the bucket when it is laid on its side. With this type of round, the drilling and loading face is free of water. The bench is easy to clean off, and there is always a sizable sump without any special effort, as one side of the shaft is always 2 to 3 feet lower than the opposite. A complete cycle takes 26 hours. Including all lost time, 3 hours are required to drill and load a round, 14 hours to muck it, and 9 hours to timber it. A mucking rate of 12.3 tons per man-shift is about average. Hand-held anvil block machines, weighing 95 pounds, are used with quarter-octogon steel and detachable bits. The timber is 10- by 10-inch fir with 2- by 10-inch fir lagging (fig. 6). Currently, the timber is being treated with copper naphthenate to prevent dry rot or fungus growth. Timber is carried a maximum of 18 to 25 feet from the face.

Main Haulage Drift

Main haulage levels are 9 by 10 feet, with a 0.5-percent grade in favor of the load. Timbering is required part of the time - 9-foot sets of round fir with 3- by 10-inch lagging are used when necessary.

In driving the drift, a crew of three men is used on all shifts. Three 3-1/2-inch automatic leyners on a jumbo are used to drill out and blast 40 seven-foot holes an 8-hour shift. An average of 93 feet is drilled per man. Mucking is done by an overhead-type machine, mucking into 3-ton Granby-type cars. The ore train is pulled by a 3-ton trolley locomotive. One man is on the motor, one mucking, and one barring down the face and helping the mucker-operator. Permanent track, ditch and pipe are carried along as the drift is driven. Each crew is capable of either mining, mucking or timbering.

Blast-Hole Drifts (7 by 7)

A mining crew usually consists of two men, using 3-1/2-inch automatic leyners on an airplane set-up (an upright bar with long universal arm). This crew will set up, drill out, and blast a 7-foot, 26- to 32-hole round in a shift. The break will vary from 5 to 6-1/2 feet, depending on the ability of the particular miner and the nature of the ground. One mucker with a mucking machine and 1-ton cars can easily muck out a round a shift. Only on very long (200- to 300-foot) trams is a battery trammer used. The floors of all these 7 by 7 drifts are cleaned to the solid and barred down when the drifts are all complete. This is necessary so that, in drilling, the holes do not have to be cased.

Drifts and Crosscuts (5 by 7)

One man with a 3-inch automatic or hand-cranked machine will set up, drill out, and blast a 5-foot, 19-hole round in one shift. Mucking may be done either by mucking machine or by hand into 1-ton cars.

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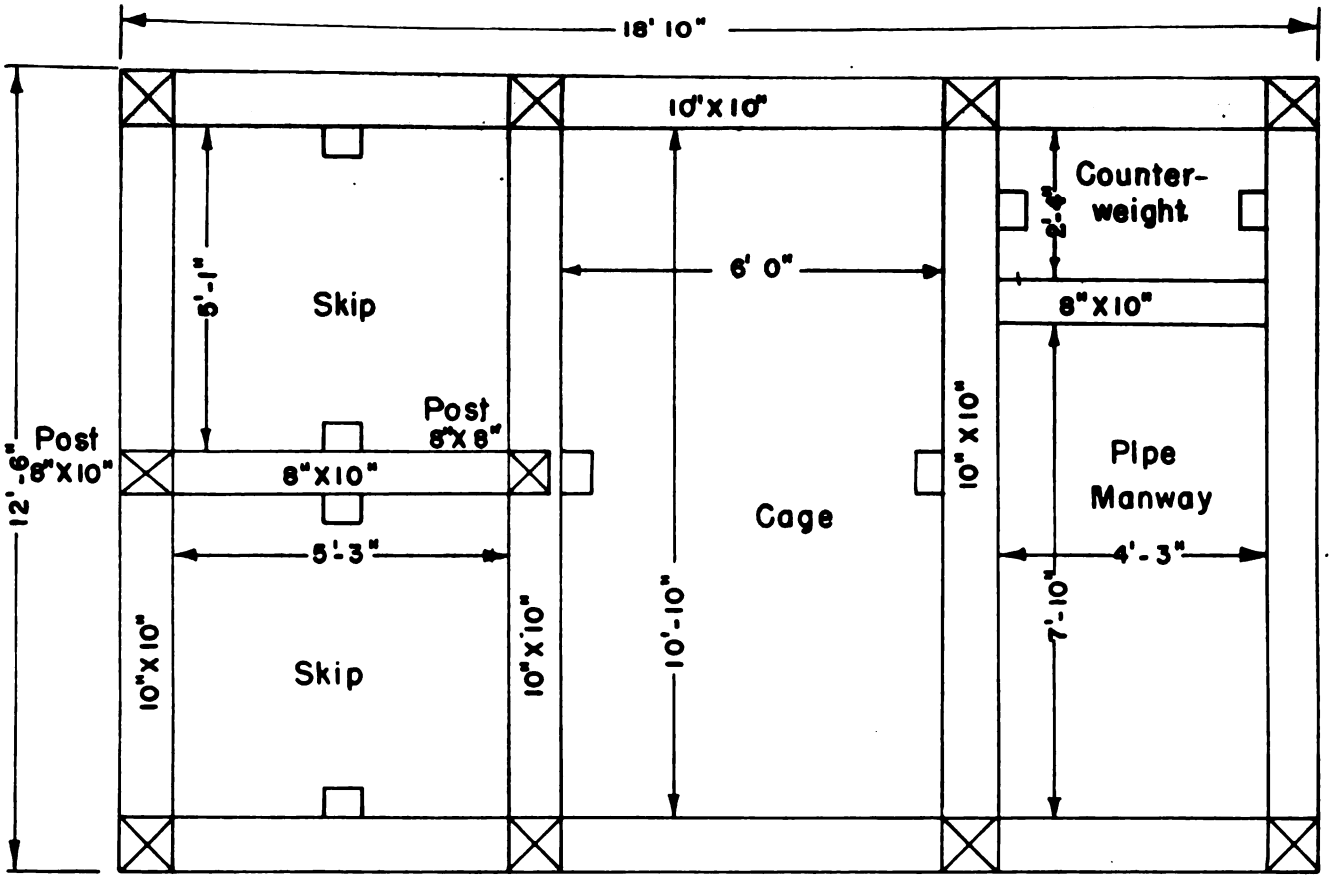

Figure 6. - No. 2 shaft timber.

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Raises

As an 8- by 8-foot raise is too large to drive full size with safety, the usual procedure is to drive a 5- by 8-foot raise, which is sideswiped later to full size. A 5- by 8-foot raise crew is usually two miners who do all their own nipping. No timbering is used in driving the raise. An iron-rung ladder is hung on the foot wall on one side, leaving space for the skip on the opposite side. An air tugger operates the skip, which is mounted on runners so that it can go up and down the raise on the rock. The drilling platform is made of 2- by 10-inch fir lagging supported by two sprags from foot to hanging wall near each rib. A two-man crew can set up, drill out, and blast a 7-foot, 22-hole round in an 8-hour shift. When the raise is completed, it is enlarged to 8- by 8-foot size by one of three methods:

1. Slabbing with a stoper from a movable platform.

2. Slabbing with three diamond-drill holes in the hanging wall of the rasie.

3. Slabbing with a drifter, using down holes.

Raises 5 by 9 feet, or 5-1/2 by 11 feet have the same size crew, but a little longer time is required to drill out the latter, as three extra holes are required.

MINING (BREAKING)

Powder-Blast Method

Undercutting a block of ground was started by driving a draw hole 5 by 6 feet in section due north at a plus 40° angle from the bulldoze chamber to intersect the foot wall of the ore. Next, east and west wings were driven on the foot wall at plus 40° to intersection, and south wings were driven at plus 40° to intersect the hanging wall. When the wings were completed, the ore body was undercut from wall to wall by stopers and drifters to form a cone above each draw point. If the undercut was extremely wide, temporary pillars were left at the hogsback between the cones. These pillars were shot just before the undercutting was completed. Service was maintained through the draw hole and wings to the east or west of the area being undercut. These undercuts were necessarily large because they had to accommodate the total muck of the first powder blast immediately above. This blast might have a burden of 20 to 25 feet or more.

Coincident with the undercutting operations, drifts, crosscuts, and pockets were being driven above the undercut for ore-breaking by powder blasting. The burden on any blast could not be more than two-thirds the width of the undercut beneath and in no case could exceed 35 feet. The powder drifts were 4 by 6 feet in section and usually were in the center of the ore. Short crosscuts of the same size were driven to the foot and hanging walls, where 3- by 4-foot pockets were driven along the walls. Accurate maps of all work done were needed so that when the powder load was calculated, the burden and position of all pockets were known. Calculations for each pocket

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in a blast were made separately by two or three members of the staff and checked by the manager. The proper load was then decided upon. The average load to the end of 1945 was 0.89 pound of powder per ton of rock. The Britannia method of calculation, which was the minimum for the Holden ground, was one case per foot of burden up to 30 feet and two cases per foot over 30 feet. Some shots would have a powder factor as low as 0.5 and others above 1.0.

A 40-percent semigelatin in large bags was primed by one-third case of 60-percent N.G. dynamite, with two No. 6 electric blasting caps wired in series. The powder was removed from the boxes and packed well into the pocket, with the primer in the center. Each pocket was back-filled with muck for the full length of the pocket and well into the crosscuts. The blasts were always shot from outside the mine. The last powder blast was shot on April 22, 1945. This brought the total to 240 shots which used 60,182 cases of explosive and produced 3,371,701 tons of broken ore. The primary powder factor was 0.892. The over-all powder factor per ton of ore drawn from stopes to the end of 1944 was 0.76 pound per ton.

Secondary blasting on the grizzlies was done by plugging all boulders that could safely be reached. The hang-ups were bombed only when washing or raking could not bring them down. Ventilation was a big problem; consequently, blasting was kept to a minimum. Secondary blasting has amounted to 0.31 pound per ton, totaling 1.07 pounds per ton produced.

The powder-blast method of mining had its disadvantages in that it shattered the walls, causing dilution. Furthermore, large blasts caused damage many feet away in the permanent workings. The large quantities of powder underground just before a blast were a definite hazard. Working quarters were very cramped for the men, both in driving and back-filling the 3- by 4-foot powder pockets. All of these reasons contributed somewhat to the change-over to diamond-drill blast holes for ore breaking.

DIAMOND-DRILL BLAST-HOLE MINING

The two variations of diamond-drill blast-hole methods to be considered are (1) the Noranda, or vertical ring drilling from drifts, and (2) the flat-hole system, or horizontal ring drilling from raises.

Noranda or Vertical-Ring Drilling (1-3/16-inch Hole)

The Noranda system of drilling vertical rings from both foot- and hanging-wall drifts was tried after several unsuccessful attempts at other systems. It was more successful than anything attempted, but further experimental work was done to compare it with the Aldermac system of drilling parallel holes from a bench. The disadvantages found in the latter were:

1. The added expense of cutting a slot.

2. The danger of working too near the stope.

3. Intermittent drilling and bench cutting.

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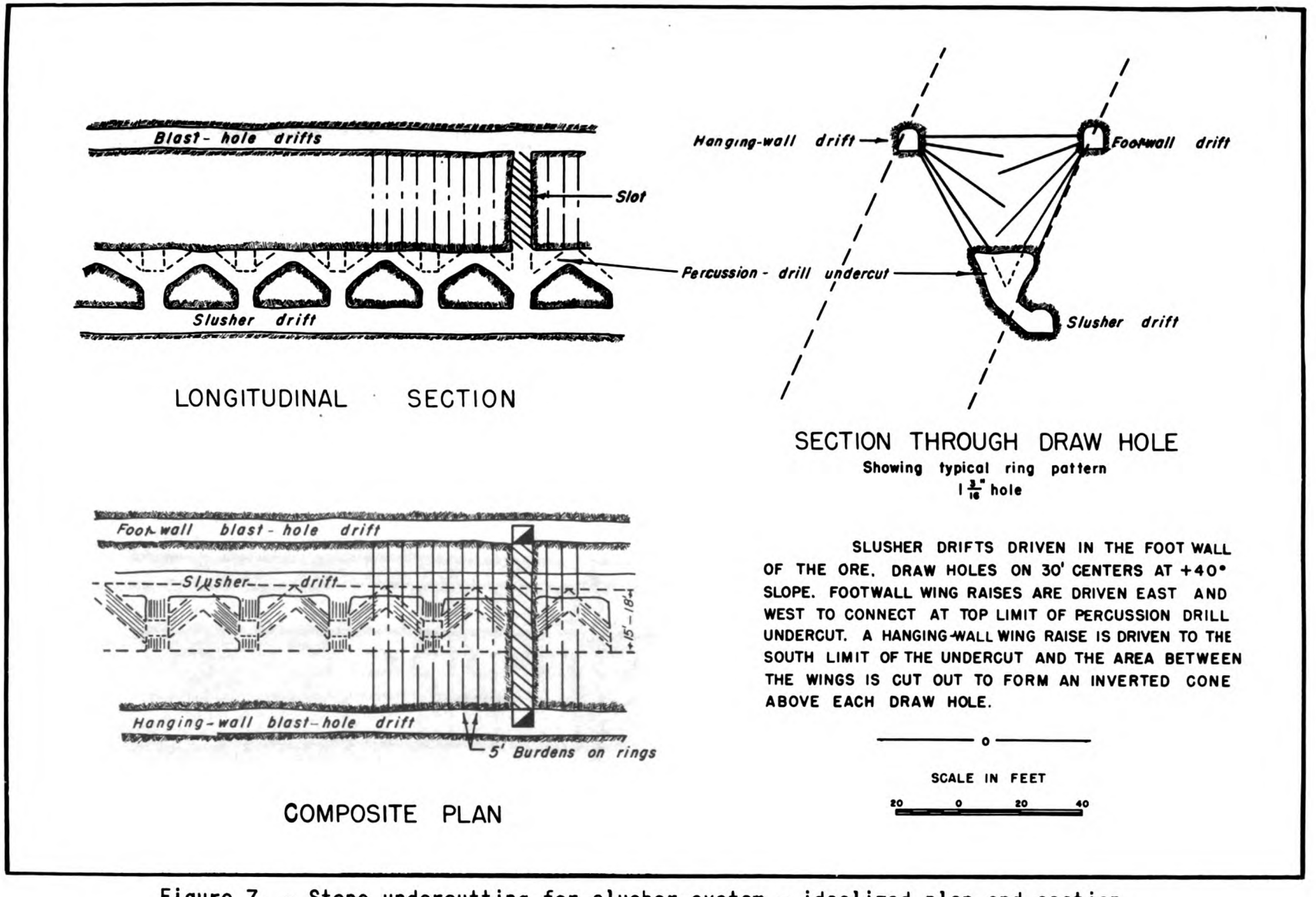
Blast - hole drifts
Slot
Slusher drift
LONGITUDINAL SECTION
Hanging-wall drift
Footwall drift
Percussion - drill undercut
Slusher drift
SECTION THROUGH DRAW HOLE
Showing typical ring pattern
1 3/16" hole
Foot-wall blast - hole drift
Slusher drift
15' - 18'+
Hanging-wall blast-hole drift
5' Burdens on rings
COMPOSITE PLAN
SLUSHER DRIFTS DRIVEN IN THE FOOT WALL OF THE ORE. DRAW HOLES ON 30' CENTERS AT +40° SLOPE. FOOTWALL WING RAISES ARE DRIVEN EAST AND WEST TO CONNECT AT TOP LIMIT OF PERCUSSION DRILL UNDERCUT. A HANGING-WALL WING RAISE IS DRIVEN TO THE SOUTH LIMIT OF THE UNDERCUT AND THE AREA BETWEEN THE WINGS IS CUT OUT TO FORM AN INVERTED CONE ABOVE EACH DRAW HOLE.
0
SCALE IN FEET
20 0 20 40

Figure 7. - Stope undercutting for slusher system - idealized plan and section.

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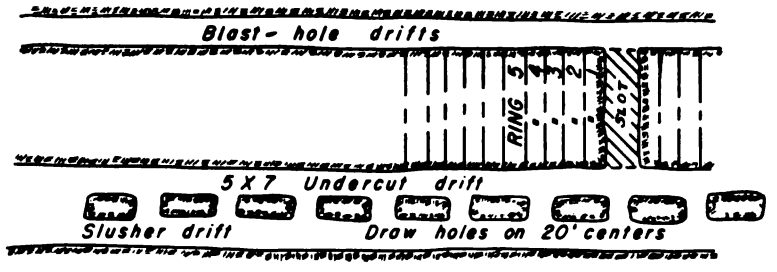
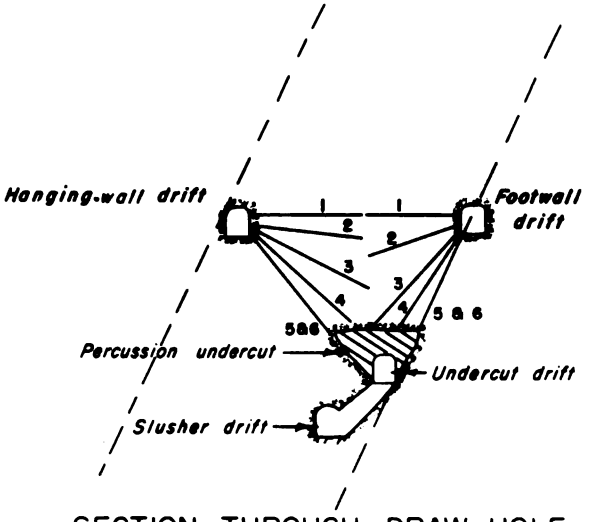


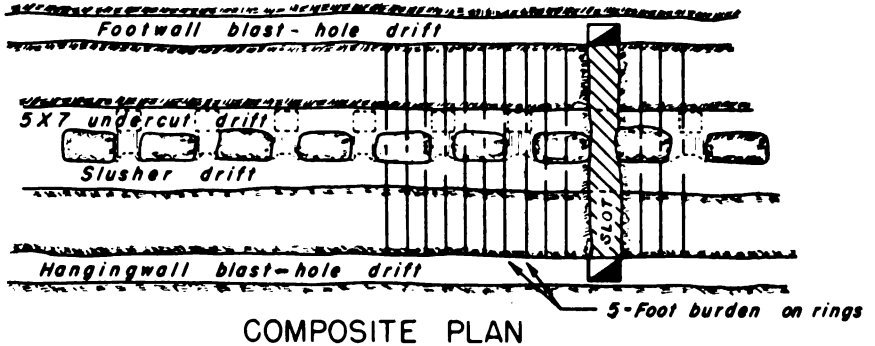
LONGITUDINAL SECTION



SECTION THROUGH DRAW HOLE

Showing typical ring pattern

1 1/16" hole



COMPOSITE PLAN

UNDERCUT DRIFTS WERE TOO SMALL AND BOTTOM OF RINGS BOOTLEGGED BADLY. WHEN THE PERCUSSION UNDERCUT (SHADED ABOVE) WAS TAKEN OUT THE RINGS BROKE VERY WELL

DRAW HOLES ON 20-FOOT CENTERS DID NOT PROVIDE ADEQUATE PILLAR SUPPORT, WHICH RESULTED IN EITHER COSTLY REPAIRS OR LOSS OF THE DRAW HOLE.

SCALE IN FEET

20 0 20 40

Figure 8. - Stope undercutting for slusher system - idealized plan and section.

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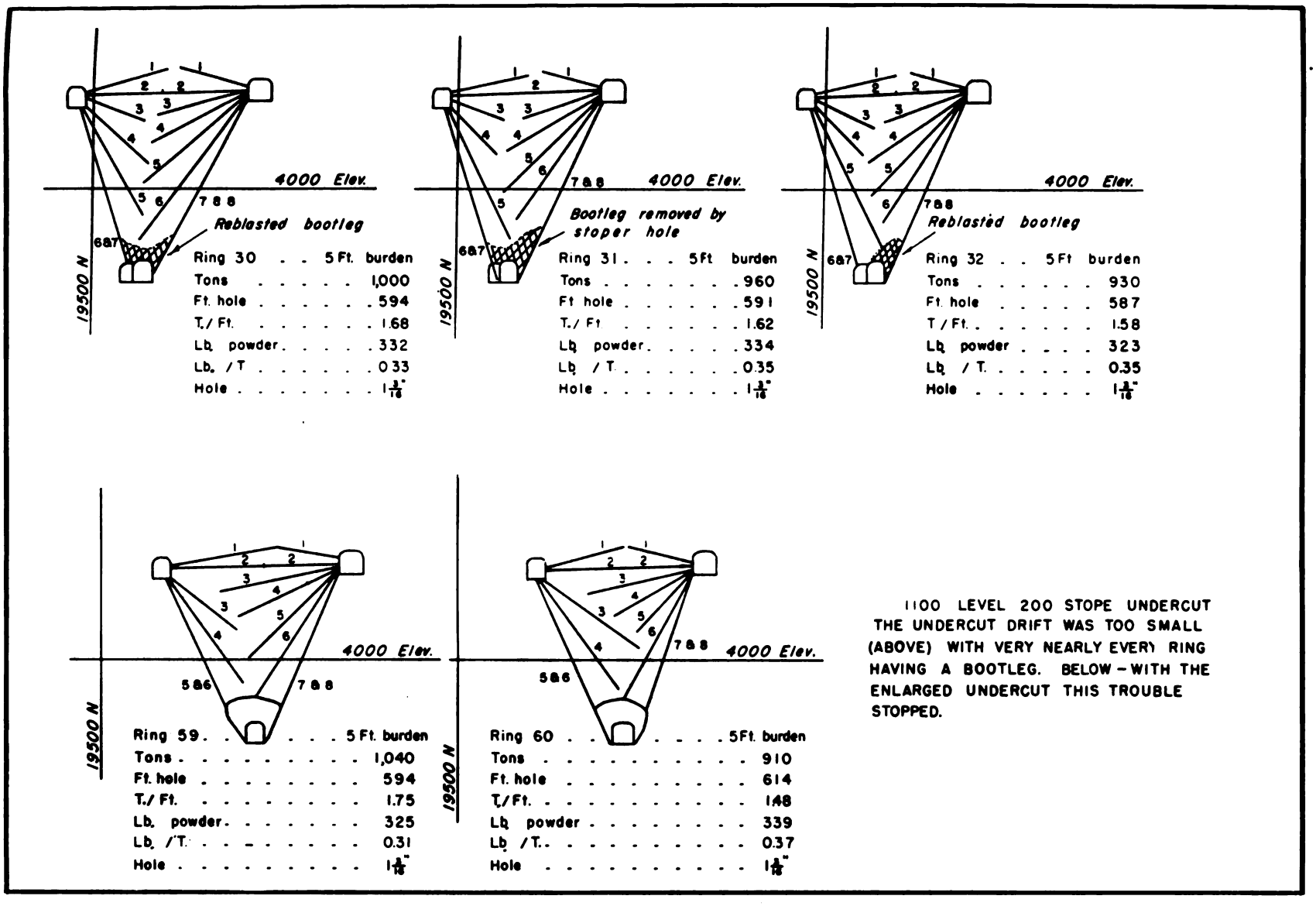
4000 Elev.
19500 N
Reblasted bootleg
Ring 30 . . 5 Ft. burden
Tons 1,000
Ft. hole 594
T./ Ft. 1.68
Lb. powder. 332
Lb. / T 0.33
Hole . . . . . 1 3/16"
Bootleg removed by stoper hole
Ring 31 . . . 5Ft burden
Tons 960
Ft hole 591
T./ Ft 1.62
Lb. powder. 334
Lb. / T. 0.35
Hole . . . . . 1 3/16"
Reblasted bootleg
Ring 32 . . 5 Ft burden
Tons 930
Ft. hole 587
T / Ft. 1.58
Lb. powder 323
Lb. / T. 0.35
Hole . . . . . 1 3/16"
Ring 59. . . . . 5 Ft. burden
Tons 1,040
Ft. hole 594
T./ Ft. 1.75
Lb. powder. 325
Lb. /T. 0.31
Hole . . . . . 1 3/16"
Ring 60 . . . . . 5Ft. burden
Tons 910
Ft. hole 614
T./Ft. 1.48
Lb. powder 339
Lb. /T. 0.37
Hole . . . . . 1 3/16"
1100 LEVEL 200 STOPE UNDERCUT
THE UNDERCUT DRIFT WAS TOO SMALL (ABOVE) WITH VERY NEARLY EVERY RING HAVING A BOOTLEG. BELOW - WITH THE ENLARGED UNDERCUT THIS TROUBLE STOPPED.

Figure 9. - Typical rings, 1100 level, 200 stope.

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4. Separate set-ups for each hole drilled.

The Aldermac system was not used after this experimental stope was completed.

The Noranda stopes in operation at the present time are the 247, 257, and 310. In preparation for the undercut, the draw holes are driven 5 by 6 feet at plus 40° on 30-foot centers out of the slusher drift, which may be in the foot wall or in the ore. East and west wings are driven on the foot wall at plus 40° to intersect. A percussion-drill undercut 6 to 7 feet high and 15 to 18 feet wide is excavated throughout the full length of the stope after the intersection of the wings (figs. 1 and 7). A 6-foot percussion-drill slot is cut from the top of the machine undercut to the level of the undercut drifts and extends from the foot to the hanging wall. While the percussion drill undercut and slot is being cut, diamond drilling is started in the undercut drifts, so that there will be some completed rings ready to blast when necessary. In that way, the drills can be operating at a safe distance from the loading and blasting.

Undercuts

Rings of holes drilled from the blast-hole drifts are laid out at right angles to the average strike of the ore at the elevation of the percussion-drill undercut and the undercut drifts themselves. This is important so as not to get an acute-angle break on one wall and an obtuse break on the other. The burden on each ring of holes can be varied from 4 to 5 feet, but the average tons broken per foot of hole is 1.8 and may go as high as 2.0. Relievers are drilled on both the foot- and hanging-wall drifts along the wall and halfway between the rings to break the walls clean and prevent bootlegs (figs. 7, 8, and 9). The foot-wall rock is harder and more difficult to break; consequently, it is favored somewhat in laying out the pattern on which the holes are to be drilled.

When blasting the rings drilled from the undercut drift or the first sublevel above, the face must be clear of all broken muck, as the burden on the ring of holes to be blasted is so near to the maximum that it will not break if the burden of the broken muck is added.

Slots

The diamond-drill slots are the tightest work done in stoping; only 0.8 ton per foot of hole drilled is obtained in an 8-foot slot. Burdens of 3 and 3-1/2 feet are used with three or four holes per row. Great care must be used in lining up the holes in a slot, as being off-line by even 1 foot at the bottom may mean that the hole will bootleg. All holes must be holed through into the crosscuts below and loaded their full length to break well. All holes are shot with instantaneous caps (see fig. 1 for plan and section of slots).

Stopes

In the 310 stope, the blast-hole drifts are so laid out that the stope can be mined in the form of a rill. Each sublevel blast-hole drift is a unit

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in itself and can be shot separately or as a full face from top to bottom. This has a decided advantage, as it permits greater flexibility for spreading the ore over more draw points. Another advantage is that more broken tonnage can be kept in the stope after the first lift above the undercut has retreated for some distance. However, the ground caves considerably under each unsupported back. This results in large boulders, which have not been shattered by blasting. In figure 1, S-N section allows for no flexibility in blasting. The whole face must be blasted at once. (See fig. 10 for lay-out of present 310 stope, which is ideal in that each set of foot and hanging drifts can be shot separately.)

In the 257 stope, holes are drilled both up and down from the first foot- and hanging-wall blast-hole drift above the undercut, making it necessary for the whole face to be shot at once. The muck from the blast piles up high against the face of the stope; and here again, as in the undercut, the muck must be drawn down from in front of the foot-wall holes before the next ring can be shot. This limits the broken-ore reserve in the stopes to practically nothing for the full life of the stope.

If the blast holes are drilled too far in advance of blasting, the holes tend to cut off in either the first 10 feet from the collar, or near the stope back. The movement of blocks of ground along fractures offsets the hole sometimes as much as one-half to three-fourths the diameter. If the hole is offset enough so that 7/8-inch loading sticks will not go down the holes, reaming is necessary. Holes will stand 6 to 8 months in most of the stopes, depending on local conditions, but beyond that there is great danger of losing the hole. Drilled-out footage should be kept at a maximum of 4 to 6 months ahead to avoid this trouble.

The same general pattern is used in all Noranda stopes to get a tonnage factor of 4 tons per foot of hole. A 6-foot burden with a reliever on the foot wall is standard practice. Holes are closer together on the foot and the hanging wall, with the widest spacings in the middle of the stope (fig. 11).

Loading

Holes are loaded with 1-1/8- by 12-inch 80-percent gelatin powder, primed with No. 8 E. B. caps with 12-foot lead wires. As many holes as possible are loaded with instantaneous caps to keep from cutting off the powder train in a delayed hole, which invariably happens when delays are used. The only place where delays are used is in the ring hole directly behind the reliever. Primers are placed 10 feet from the collar of the hole, and the hole is loaded with powder up to within 2 to 7 feet of the collar, where the holes are close together. Loading sticks are 1 inch in diameter and 6 feet in length and are equipped with a lock-hook copper connection.

Normally a good loader can load an average of 200 to 300 feet per shift. Under very good conditions he can load very nearly double that amount. The powder is so near the size of the hole that tamping often does not greatly increase the amount of powder loaded. Thirteen 1-1/8- by 12-inch "redi-split" cartridges can be tamped at once to fill 12 feet of hole. This quantity fills 96 percent of the calculated volume of the hole. Slitting the paper cartridges

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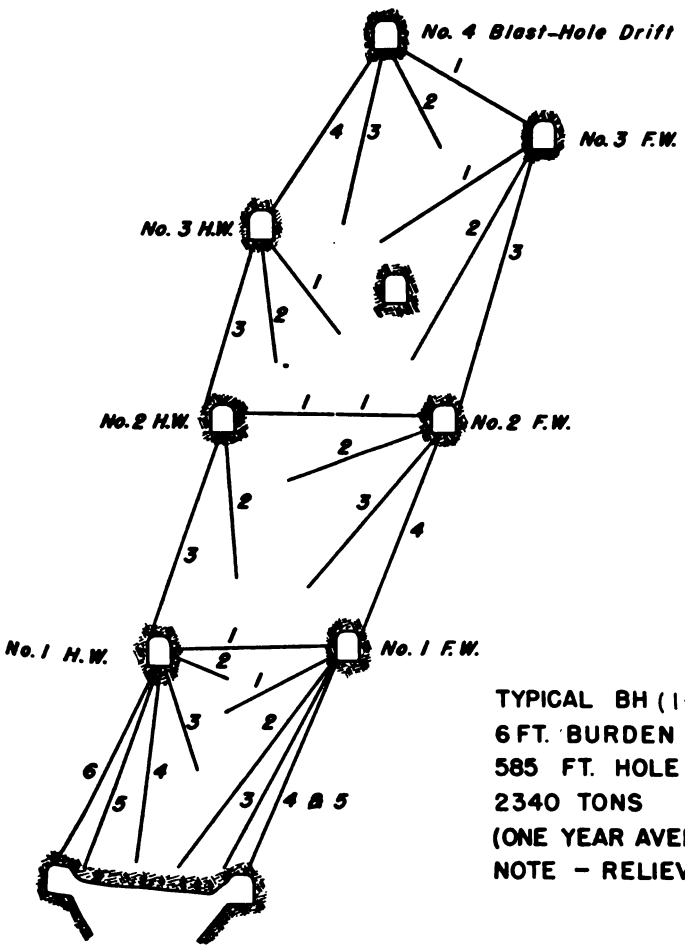

AX RING ($1\frac{10}{16}$")
8 FT. BURDEN
815 FT. HOLE
8194 TONS 10.05 TONS / FT.
0.13 LB / TON. *
* EXPECTED POWDER FACTOR
NOTE – NO RELIEVERS

TYPICAL BH ($1\frac{5}{16}$") RING
6 FT. BURDEN
585 FT. HOLE
2340 TONS 4 TONS / FT.
(ONE YEAR AVERAGE) 0.12 LB. POWDER / TON
NOTE – RELIEVERS ON FOOTWALL

RING PATTERNS OF AX AND BH HOLES

20 0 20 40 60
SCALE IN FEET

Figure 10. - Typical rings, 1775 level, 310 stope.

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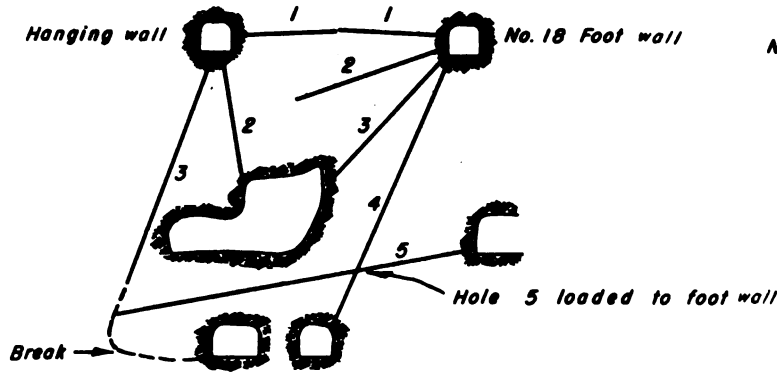
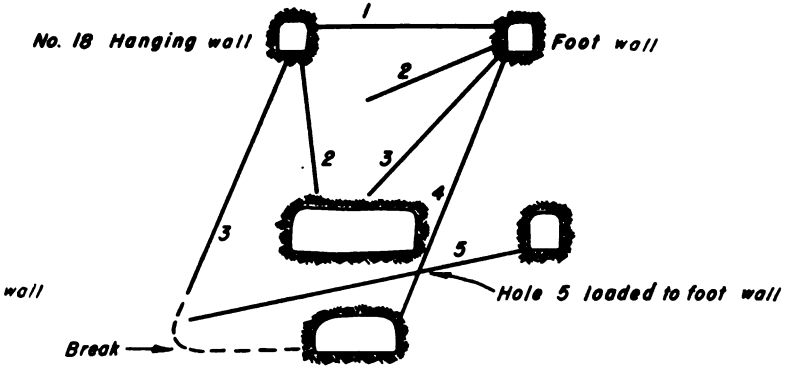


RING 98

7 FT. BURDEN
401 FT. HOLE (AX-1 15/16 IN.)
2691 TONS 6.7 TONS / FT.
500 LB. POWDER 0.185 LB. / TON



RING 99

7 FT. BURDEN
394 FT. HOLE (AX-1 15/16 IN.)
2566 TONS 6.5 TONS / FT.
450 LB. POWDER 0.175 LB. / TON

BREAKING WAS MADE MORE DIFFICULT BECAUSE OF A SERVICE RAISE AND A HAULAGE DRIFT IN THE BLOCK OF GROUND TO BE MINED.

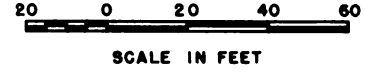


Figure 11. - Rings 98 and 99, 1775 level, 247 stope.

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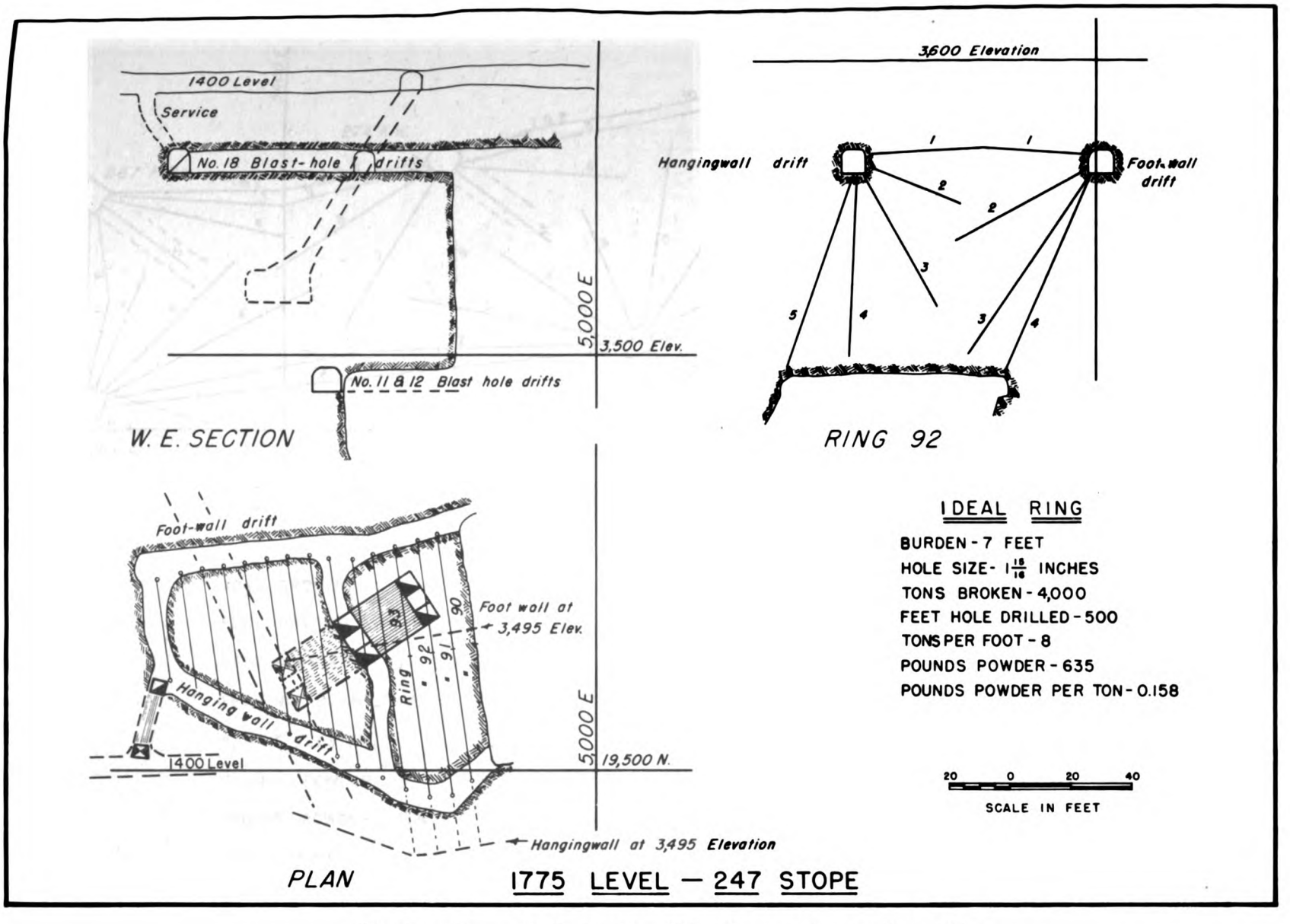

Figure 12. - 1775 level, 247 stope; plan and section.

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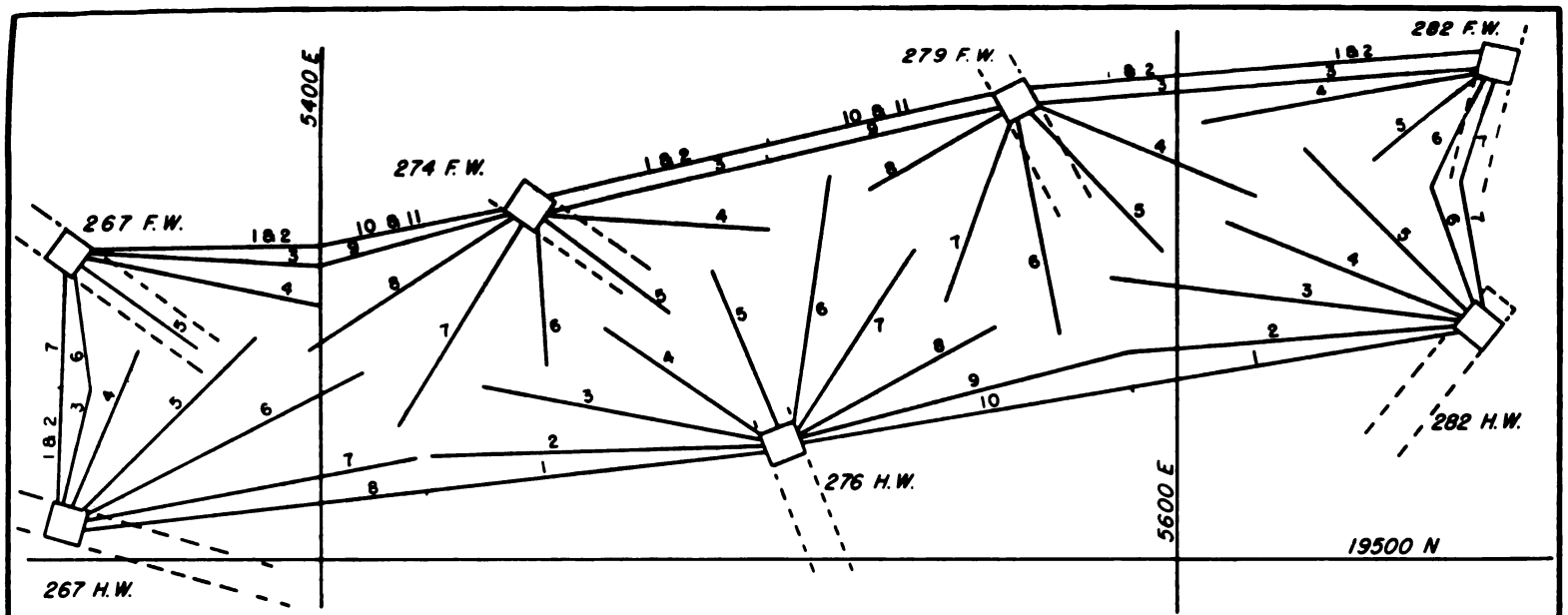

RING 10 ELEVATION 3412 BURDEN 6'

	280 STOPE-RING 10	280 STOPE (25 RINGS)
TONS BROKEN	12,785	207,715
FEET DRILLED	3,125	53,258
FEET BLASTED	3,125	42,151
FEET NOT BLASTED *	——	11,107
% NOT BLASTED	——	20.8 %
POUNDS POWDER	1,625	22,553
POUNDS POWDER / TON	0.13	0.109
TONS / FOOT	4.09	3.09

* Holes that caved into the stope or were cut off and could not be loaded.

Figure 13. - Ring 10, 280 stope.

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with a knife before tamping actually decreased the amount of powder that can be tamped into a hole, whether tamped after each stick or after every 10 sticks.

AX Blast Holes (1-15/16-Inch)

A small experimental stope, using AX holes (1-15/16-inch) for ore breaking, seems to have some advantages over the 1-3/16-inch size in that:

1. The direct cost (drilling and loading) per ton of ore broken can be kept the same by doubling the tons broken per foot of hole.

2. The primary powder factor can be increased by 40 percent to help fragmentation with only the added cost of powder.

3. Relievers are not necessary with burdens up to 8 feet, further aiding fragmentation.

4. The powder factor can be kept the same as with the smaller holes, and the tons broken per foot can be increased so as to materially reduce the primary breaking cost. (See figs. 10, 11, and 12 for typical patterns of AX holes.)

The Noranda stopes are very safe for the drillers and loaders working in them, as all work is confined to the 7- by 7-foot drifts. No one is ever required or allowed to go into the open stopes. Loading is carried one ring ahead at all times, so loaders are always working at a minimum of 6 feet from the open stopes. Safety stulls are kept at all blasting faces to prevent anyone from falling or walking into the open stope. The trend is toward electric machines for blast-hole work, but it is very important to take every precaution to guard against stray currents where drilling and blasting are in close proximity. Electric machines tend to be very heavy and awkward for the horsepower produced and not nearly as flexible as air machines to date.

Horizontal-Ring Drilling (1-3/16-Inch Hole)

The 280 stope was run as a flat-hole stope for its full length from the undercut up to the 1600 level. Horizontal holes were drilled from four foot-wall and three hanging-wall raises with 6-foot burden and a reliever hole 3 feet below each foot-wall ring. The hanging wall was only slightly favored with close spacing of holes (fig. 13).

All drilling was done on a horizontal bar set up on a movable platform and was completed to the 1600 level, a vertical distance of 80 feet, before any blasting was done. From the 1600 to the 1500 levels, only the west half was mined from two foot-wall and two hanging-wall raises in the same manner as the lower portion. Loading and blasting procedure was nearly the same as in the Noranda stopes - all holes except the foot-wall ring hole were primed with instantaneous electric caps. A ring of holes was loaded just before it was to be blasted to keep to a minimum the quantity of powder that would fall into the stope when the back caved.

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Advantages of this system are:

1. No slot is required.

2. Broken ore is over all draw holes at all times.

3. Tonnage broken in stopes can be kept at a maximum.

Disadvantages are:

1. There is excessive arching after each shot, losing drilled-out footage up to 25 percent.
2. Drilling and loading are intermittent, unless drilling is carried far ahead.
3. There is some danger working in raises, even with protective bulkheads and movable platforms.

Drilling

All drilling and loading are contracted. The 1-3/16-inch holes are drilled with plug bits and the 1-15/16-inch holes with thin-wall core bits, using both compressed-air and electric machines. Footage per machine-shift on 1-3/16-inch hole is about 40; footage per man-shift is 26, including the shift bosses but not the foreman. The average for the AX hole is 25 feet per machine-shift.

The diamond-drill engineer is responsible for laying out the pattern of the holes in the ring and marking all rings underground. Work sheets are made for the drillers; these sheets give all required data for drilling each ring, including length, bearing, and dip of each hole. Similar sheets are made for loading; these give length, delay, and powder to be used. A drilling and loading sheet is kept for each ring as a part of a permanent record.

ORE DRAWING

Slusher Mucking Equipment

All slusher hoists are 50-hp. double-drum units. Six of them have a 6,600-pound rope pull at a speed of 250 feet per minute on the main line and 333 feet per minute on the haul back. Two units have a rope pull of 7,200 pounds with 225 feet per minute rope speed on both main line and haul back.

The buckets in use are the hoe type, weighing approximately 3,000 pounds The cable is seven-eighths inch.

The most commonly used splice is a double "Molly Hogan", which forms a square knot in the cable. It is very rapid and can be made by the least experienced operator. The main block is 18 inches in diameter, and the idler blocks are 8 inches in diameter; each has a large throat to accommodate the large splice.

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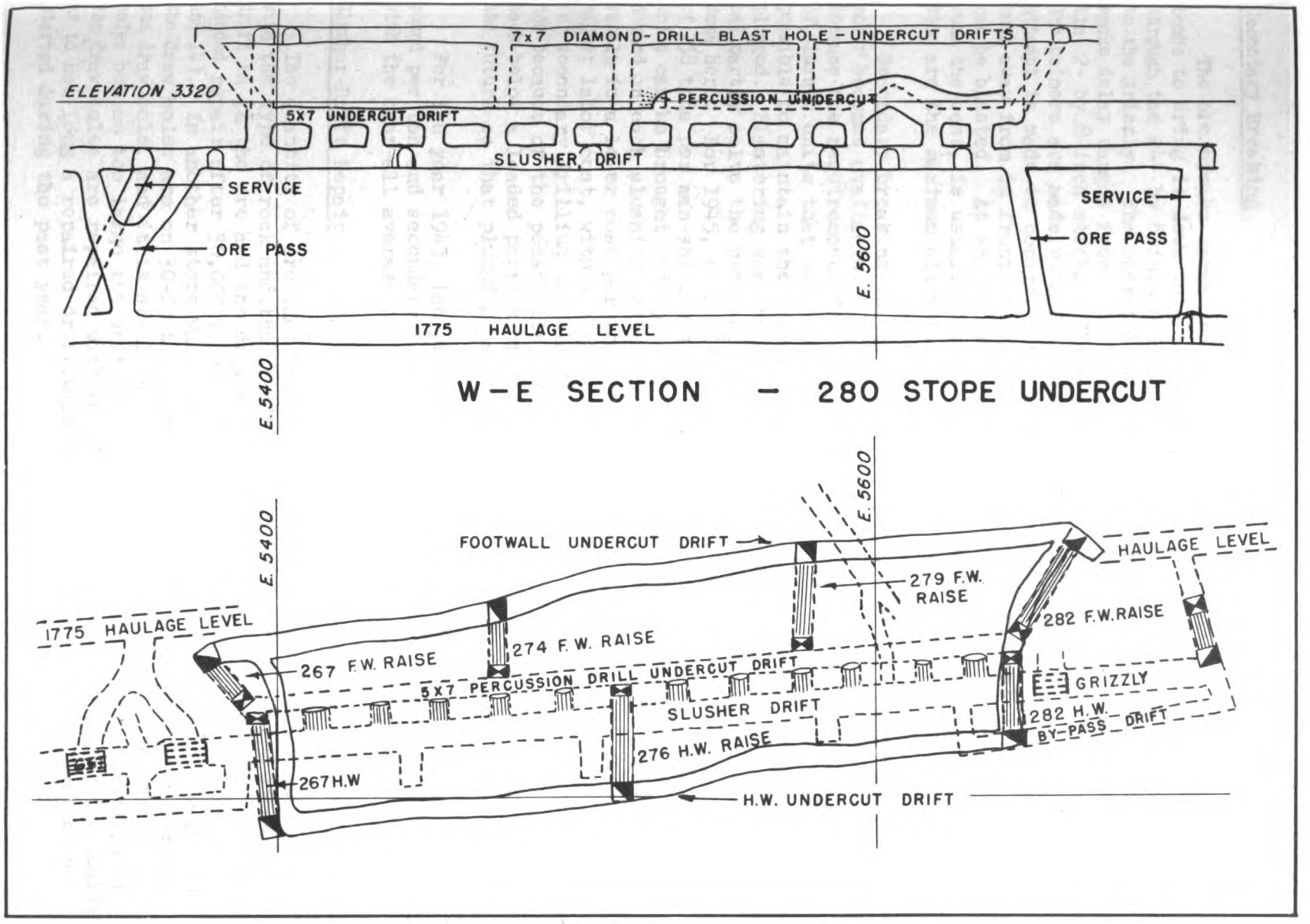

Figure 14. - Plan, 280 stope undercut.

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Secondary Breaking

The ore breaks large and hangs up in the draw holes, requiring large bombs to bring it down into the slusher drifts. Any rocks too large to go through the 24- by 6-inch grizzly openings are plastered before being scraped to the grizzly. The operator is allowed only three shots at one time. The smoke delay ranges from 1 to 5 minutes. The powder used is 40 percent gelatin, 2- by 8-inch sticks. These sticks make up very easily into plasters. The primers are made with No. 6 manasite caps with 3- and 6-foot fuses. No attempt is made to control the draw, except when it is urgent to clear the muck away from in front of the blasting face so that another ring of holes can be blasted. At any other time, the nearest draw hole, or the one that runs the best, is usually drawn. For safety reasons, three shots at one time are the maximum allowed.

Secondary breaking in the slusher drifts can be improved when more manpower becomes available. It has been abnormally high during the war years, because the requirements for a high tonnage per man unit limited the number of slusher units that could be put into operation. It would have been impossible to maintain the tonnage desired if all the oversize rocks had to be plugged. Plastering was the only possibility at the time. Now shaped charges may partly solve the problem, although no work with shaped charges has been done here. For 1945, with six units in operation most of the time, an average of 158 tons per man-shift was obtained with one man on each unit. If more units can be brought into production in the future, there will be a smaller demand on each slushing unit, and more plugging can be done. This should result in a lower cost per ton drawn. The best results might show a slightly higher labor cost, with a large cut in powder consumption. Using pluggers for secondary drilling and blasting of the blast hole muck is very dangerous now because of the possibility of powder in the muck that sloughs off the back below a loaded portion of the stope. This danger may be eliminated in the future so that plugging may again be used.

For the year 1945, the primary powder usage in blast-hole stopes was 0.19 pound per ton and secondary 1.01 pounds - a total of 1.20 pounds, as compared with the over-all average of 1.07 pounds for powder blasting to date.

Slusher-Drift Repair

The quantity of ore that can be drawn from a draw hole varies somewhat with the type of rock and the spacing of the draw holes. Where the slusher drift is in the ore and the draw holes are on 20-foot centers, the draw holes needed repair after 30,000 to 40,000 tons had passed through them (figs. 8 and 14). In another stope where the slusher drift is in the foot wall and the draw holes are on 30-foot centers, 90,000 tons have been drawn through one draw hole, and it is still in good condition (fig. 7). When the draw holes become too large the unit is shut down, and the drift and collars of the draw holes are repaired with reinforced concrete. No data are available as to how long a repaired drift will stand up. Concreting has just been started during the past year.

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UNDERGROUND HAULAGE

Chutes

A standard chute that will handle large volumes of coarse muck has been developed. The chute is of heavy timber construction, with 1-inch mild-steel bottom liners, which will last the life of the chute. Each chute has two air cylinders - one to operate the undercut arc gate and the other to operate the four 1,600-pound fingers. This gives the chute puller very close control on the draw. A spill guard, hinged on either side of the chute, rests on the arc gate; and, as the gate is lowered, the guard drops down to the top of the side of the car opposite the chute. This arrangement prevents excessive spill, (fig. 15).

Haulage, Dumps, and Pockets

	1500 Level	1775 Level
Locomotive type	Trolley	Trolley
Weight	13-ton	8-ton
Voltage	250 volts D. C.	250 volts D. C.
Track gage	30-inch	30-inch
Rail	60-pound	40-pound
No. cars to train	10	12
Cubic feet capacity	170	84
Tons per car	7.5-7.8	3.0-3.3
Length of haul	2,965'	640'-1,920'
Tons per hour	140	190

The smaller cars are used on the lower level for two reasons:

1. Smaller haulage level.

2. Cars can be put on the cage and hauled up the shaft and outside for repair, while the large 7-ton cars have to be repaired underground.

The underground dumping ramp for the 3-ton cars was fabricated in the shop in three sections, taken underground and erected. Its main advantage is that the few moving parts on it cause little wear. It is operated by a solenoid switch which causes it to move into position just before the train goes through to dump and then move back out of position before the train goes back through empty. An oil-retarding cylinder lowers the speed of movement of the ramp, thus reducing the shock.

The ore pocket above the crusher has a 1,500-ton capacity. It was enlarged from a 5- by 8-foot raise by diamond-drill blast holes. This was done during the time when the labor shortage was most critical. It did save a great deal of manpower and was quite reasonable in cost. A tonnage factor of 0.59 ton per foot of hole was obtained (fig. 16).

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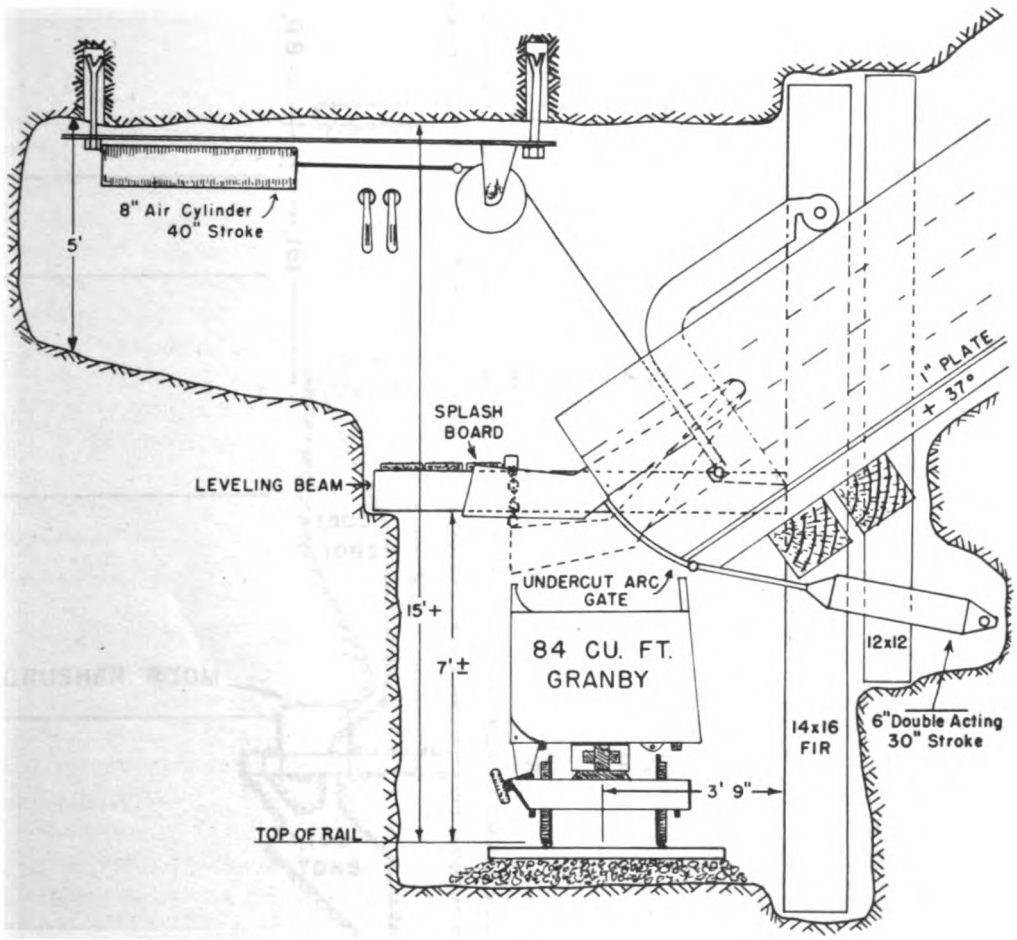

Figure 15. - Loading station and air-operated chute gate.

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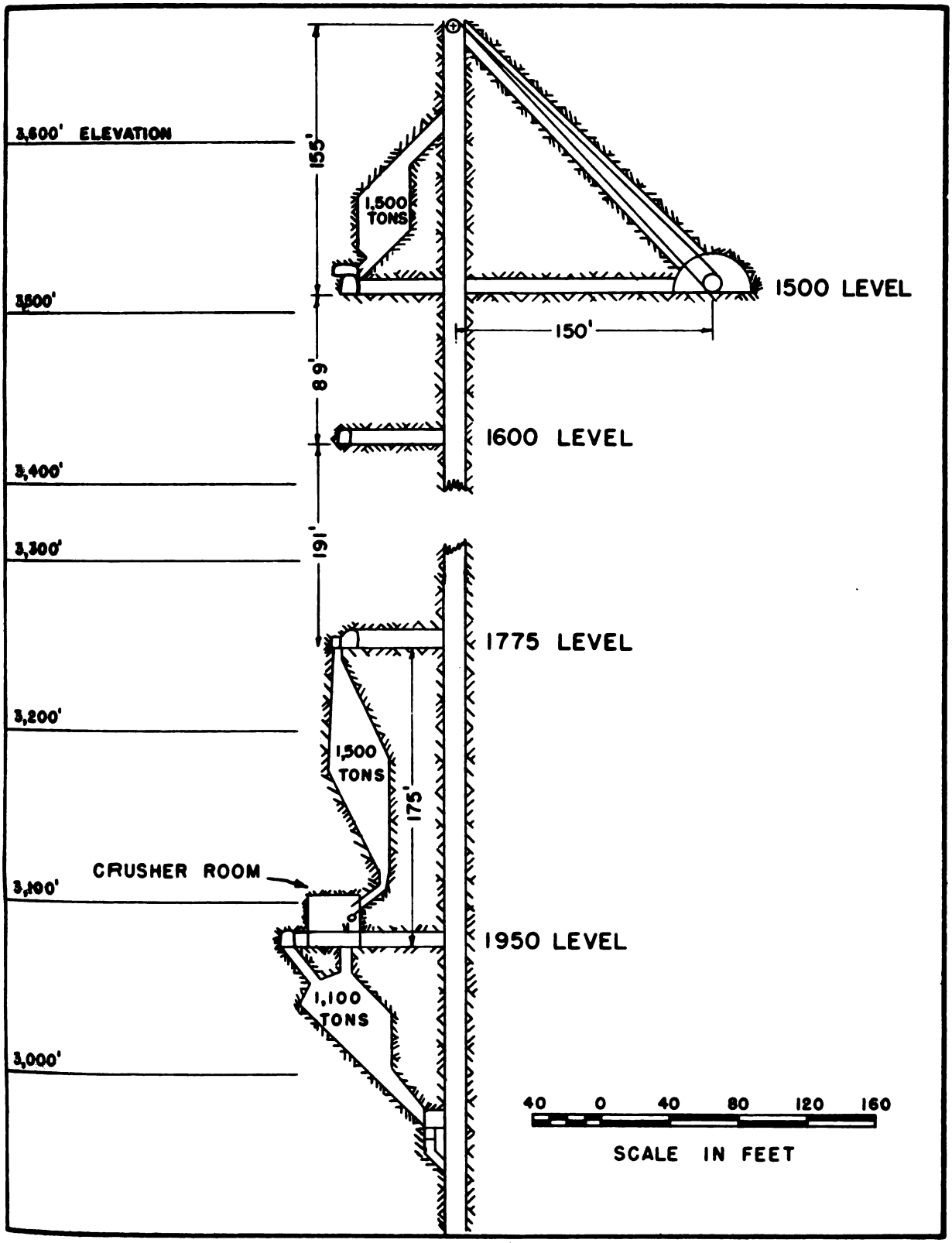

Figure 16. - No. 2 shaft hoist room, crusher and skip pockets.

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HOISTING

From the coarse-ore pocket, the ore is crushed to minus 6 inches by a 36- by 48-inch jaw crusher. It then falls into an 1,100-ton ore pocket and thence into the chutes in the No. 2 shaft. The ore is hoisted by two 4-ton skips, hoisting in balance at the rate of 185 tons per hour. The pocket on the 1500 level holds 1,500 tons of ore, which gives a total underground storage of 4,100 tons in the No. 2 shaft pockets alone, plus an additional 600 tons in the short raises directly below the slusher drifts. The ore hoist is a double-drum hoist, powered by a 600-hp. motor with Lilly controller. It has a recorder which shows the position of the skip at all times during the day and is also very convenient for count. Rope is 6 by 19 best plow steel lang-lay, 1-1/8-inch diameter.

PERCENTAGE OF EXTRACTION

In the first years of operation, the ore bodies as laid out were calculated as to tonnage and grade, no allowance being made for dilution of low-grade material or waste. As the mining of the blocks progressed, it became apparent that a greater tonnage was being drawn than had been broken. The hanging wall did slough to a great extent, varying with the size of the stope. As it was an assay hanging wall, the dilution was not waste, but low-grade millable ore, with no mining cost charged to it - only ore drawing, hauling, and milling. When a block was completely mined, it was found that the tonnage was greater, the average assay was lower, and the total copper and gold units recovered were higher than was originally calculated. This dilution has averaged 13.6 percent of the total tons drawn to date.

In later years, when a block of ore has been calculated, the grade has been corrected downward by 10 percent, as described under Methods of Sampling and Estimation of Tonnage and Values, but the expected tonnage is unchanged.

Less dilution is expected with the diamond-drill blast-hole system because of the relatively light blasting along the walls. In one stope that was completely mined by this method, the overbreak was nil; but it is expected that, when larger stopes are opened, there will be some slough. The 257 stope is sloughing somewhat already, after only 39 percent of the expected tonnage has been blasted.

At greater depth, more care will be necessary in mine planning to take care of such problems as pillars for support, filling of mined areas, and other related problems.

WAGE AND BONUS SYSTEMS

The bonus system was first used in 1937, 1938, and up until July 1, 1939, when it was discontinued. However, in August 1942, it was again necessary to start the bonus system, as the manpower shortage would not allow operations to continue on the day's pay basis. The bonus prices were first tried on the development headings. These were based on current costs, which enabled a good miner or mucker to make a bonus above day's pay, according to his ability. It was immediately successful, and the plan was expanded to include all

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volume work measured in cubic feet, timbering measured by the linear foot or set, and ore drawing and ore hauling measured by the ton drawn and delivered to the mill. The men benefitted by the extra earnings, and the company benefitted by keeping the operation going at a time when it was impossible to compete with the high wages in the war industries. At the same time the costs were kept at a normal level. (See fig. 17 for average effective mine crew, 1941-45.)

The bonus prices on all underground work-driving drifts and raises, timbering, ore drawing, and ore hauling - were calculated by the mine superintendent, foreman, and production bonus engineer, basing the calculations on normal operations. All headings are a standard price wherever possible, with variations based on length of tram, service, and mucking equipment to be used. The prices are the same in ore or waste.

When a miner is given a heading on bonus, the mine foreman makes out a bonus memo in triplicate - he keeps one, and sends one each to the timekeeper and to the production bonus engineer. The miner is informed of the price to be paid, the number of men on the bonus, specifications as to size of heading, and service provided. The engineer measures the footage advance after each week, and the timekeeper keeps time on a special bonus time sheet for each heading in the mine. The weekly advance and price are put on each bonus sheet at the end of the week and checked by the mine superintendent. The miner receives a copy showing his footage advance, price, base pay, base bonus, and bonus overtime for the week. All slusher-drift repair, concrete bulkheads, ventilation doors, and track and pipe salvage are put on bonus.

For 1945, an average of 43.2 percent of the underground crew was on bonus.

VENTILATION - NATURAL AND MECHANICAL

It is very important that ventilation be good in order to get the high tonnage in the slusher drifts where such a great amount of blasting is required. The smoke delay must be at a minimum, so that slushing can be started as soon after a blast as possible.

A fan of 110,000 c.f.m. capacity at 9-inch water pressure discharges air from the exhaust system at the west end of the mine. A booster fan of 50,000 cubic feet capacity at 2-inch water pressure is near the east end of the main haulage level at the top of the downcast ventilation raise system to the lower levels. The booster fan was necessary to equalize the volume of air to each slusher unit. Otherwise the stopes nearest the exhaust fan tends to draw more than their allotted volume of air. The minimum volume that each slushing unit should have is 10,000 cubic feet of free air per minute.

The source of the fresh air changes with the seasons. In the summer months, the greater part of the air is drawn through the 1500 haulage level and from the ventilation raises from the upperworkings at the east end of the mine. During the winter months, the 1500 haulage level is blocked off by double doors to prevent the level from freezing. Then all the air comes from the 550 and 800 levels through the old stopes and ventilation raises. This

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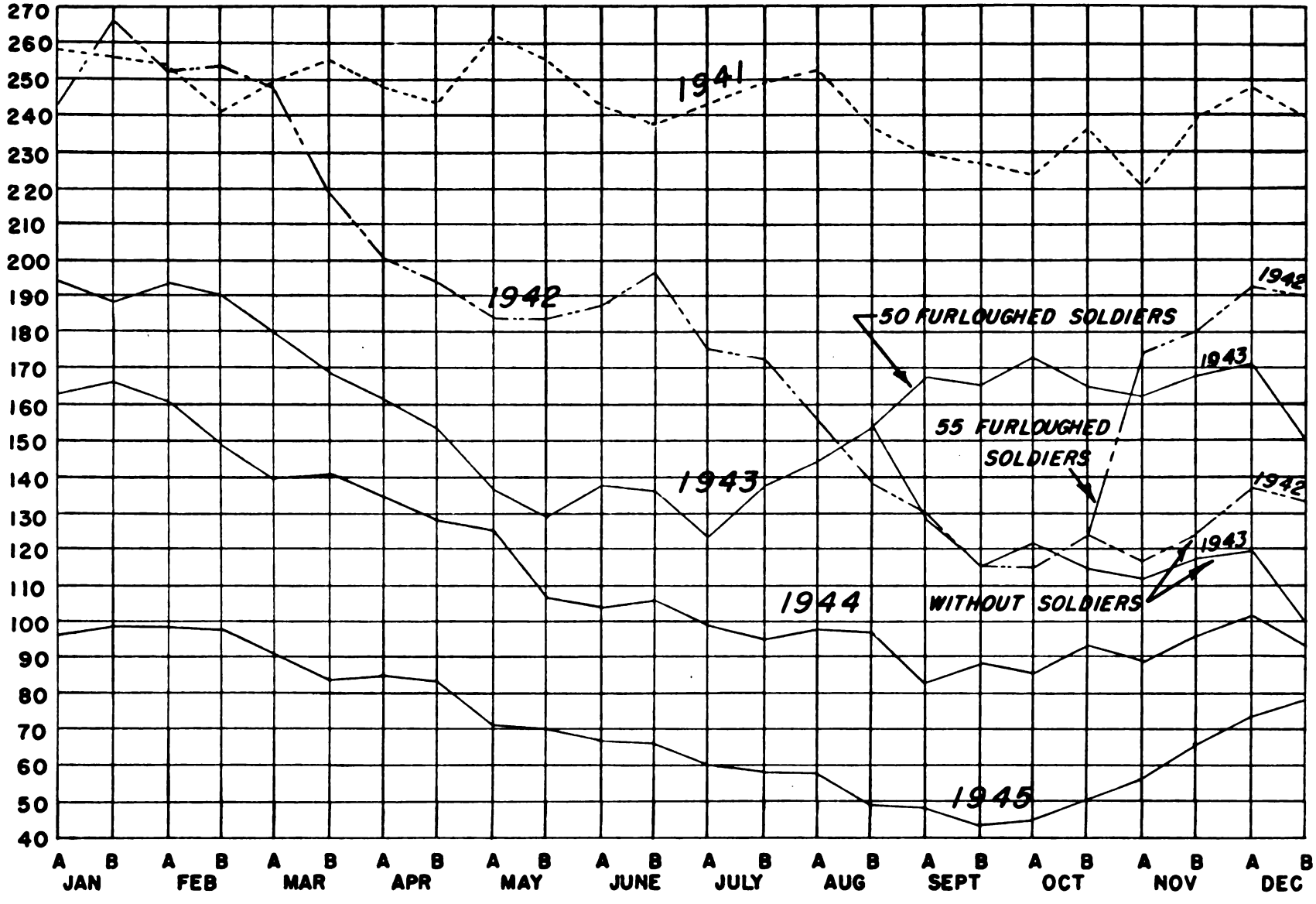

Figure 17. - Average effective mine crew, 1941-45.

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makes positive control more difficult in the winter. The main air passes for both downcast and exhaust are 5-1/2 by 11 feet at each end of the mine. The 1775 haulage level is 8 by 9 in section. The fresh air flows along it and is distributed up 6- by 6-foot raises to the slusher drifts, past the slusher operator and the draw holes, into 6- by 6-foot exhaust raises which have control doors at the slusher level. The 6 by 6 raises collect into the 5 by 7 intermediate level over a distance of 1,400 feet, go up a 5-1/2 by 11 raise to the 1500 level, and through 1,754 feet of 7 by 8 untimbered drift and 396 feet of 7 by 7 timbered drift, lagged smoothly on the insides.

Secondary fans, both air-driven and electric, in sizes from 5 to 15 hp., are used in development headings. The ventilation pipe is 15-1/4 inches diameter of 16 gage black steel. The tubing is 12-inch diameter, made of DuPont No. 1A cloth, with double suspension seam.

FIRE HAZARDS

The only fire hazards in the mine are the two shafts, one of which is kept wet as much as possible to keep the hazard down. Smoking on the shaft stations is prohibited. Concrete and steel fire doors have been installed on all shaft stations to confine and smother any fire that may start. Fire hose and nozzle are kept ready for any emergency in the No. 2 shaft station, 1500 level.

The inclined shaft is not as well-protected as the vertical shaft; but there is much less timbering, and the foot wall plates and dividers are always wet and could not burn. There is very little timbering in the drifts that might burn, and the ore does not have enough sulfide content to burn. There are three escapeways from the lower levels of the mine - two shafts and a ventilation raise. Instructions are posted underground at the shaft stations stating what to do in case of fire.

SAFETY METHODS

A full-time safety engineer is responsible for all mine safety underground. Each month a safety committee, consisting of the safety engineer, two miners, and a member of the engineering staff, inspects all parts of the mine and reports on all unsafe places and practices. The State mine inspector also makes periodical visits to the property and is accompanied by part of the mine safety committee to all parts of the mine. The mine foreman carries out the recommendations of the safety committee and inspector. Every 2 weeks the safety dogs in the No. 2 shaft are checked to see that they are in good working order. The hoisting ropes are kept well-lubricated and the ends cut off periodically to keep them in the best possible condition.

Each year, first-aid instructions are given 4 hours a week for a month, or 16 hours total. After this instruction, the class is tested by the Bureau of Mines first-aid field man, and first-aid certificates are given to those who pass the examination.

Miscellaneous data pertinent to the drilling and mining operations at Holden are shown in tables 1, 2, 3, 4, and 5 appended to this report. Also in the appendix are graphic charts (figs. 18 to 24), showing the record of man-hours and powder used per ton of ore drawn from various stopes. These tables and charts are self-explanatory.

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TABLE 1. - Rock-drill air consumption and drilling-speed tests

No. of drills tested	Machine - type	Piston size, in.	Ave. free air per inch drilled, cu.ft.	Drilled per min., in.	Air/min., cu.ft.	Working pressure, lb./sq.in.
19	"A" - drifter	3-1/2	25.1	8.5	188	76.5
3	"B" - drifter	3-1/2	25.6	8.3	205	75.5
18	"C" - stoper	3-1/8	35.0	6.1	196	73.0
1	"D" - stoper	2-7/8	17.7	9.5	150.5	84.0
1	"E" - stoper	-	20.4	9.4	186	79.5

"B" used in haulage drifts in very hard ground. Other machines are in general and miscellaneous use.

TABLE 2. - Diamond-drill air consumption and drilling speed

Drill No.	Static air pressure, lb./sq.in.	Working air pressure, lb./sq.in.	Air per min. corrected, cu.ft.	Speed of drilling, in./min.	Air per inch drilled, cu.ft.
"A" 1/	85	81	182.6	2.9	63.0
"B" 1/	88	87	164.9	3.5	47.1
"C" 1/	83	80	131.2	3.6	36.5
"D" 1/	84	80	131.8	4.7	28.0
Averages	85	82	150.1	3.9	43.8
"E" 2/	80	79	162.7	2.9	56.1

1/ 1-3/16-inch hole. Plugged bit.
2/ Core drilling (EXE).

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TABLE 3. - Miscellaneous data sheet

	1939	1940	1941	1942	1943	1944	1945	Total to Jan. 1, 1946 1/
Total ore drawn.....tons	597,787	587,729	685,803	715,806	720,240	756,331	618,927	5,157,503
Total ore broken..... do.	501,867	507,728	637,461	602,789	535,897	604,308	629,498	4,455,066
Current dev. footage.....	6,650	11,086	12,581	10,008	8,921	3,766	2,566	61,924
Stope dev. footage.....	6,654	12,001	12,923	9,165	12,452	12,464	2,674	73,060
Exploration.....	5,312	6,785	5,135	1,232	-	-	-	18,855
Total footage.....	18,616	29,872	30,639	20,405	21,373	16,230	5,240	153,839
Powder dev. footage.....lb.	352,950	577,046	569,800	404,950	388,000	343,450	100,850	3,159,746
Powder per foot of dev. .....do.	18.95	19.32	23.14	19.81	18.14	21.16	19.24	19.88
Powder ore breaking.....do.	320,525	574,060	538,500	437,800	475,395	469,450	176,300	3,178,130
Powder per ton broken.....do.	0.638	1.20	0.84	0.727	0.89	0.68	0.28	0.69
Powder ore drawing.....do.	132,549	101,771	106,900	170,250	345,425	442,450	566,000	1,930,325
Powder per ton drawn.....do.	0.239	0.147	0.16	0.242	0.465	0.59	0.91	0.37
Blasting caps used.....	-	67,636	81,560	83,511	91,494	104,860	127,496	693,769
Electric caps used.....	-	178,879	198,567	112,046	117,518	91,585	39,778	917,960
Conventional steel sharp.....	-	743,758	257,254	5,148	7,713	-	13,788	1,781,272
Detachable bits sharp.....	-	-	372,395	323,395	105,111	-	2,036	870,704
New pits used, "A" type.....	-	-	-	-	223,700	287,450	90,202	601,352
New bits used, "B" type.....	-	-	93,785	97,989	38,700	-	-	247,720
Total drilling hours.....	44,083	75,790	76,601	47,181	41,644	29,394	9,859	359,375
Air compression.....kw.-hr.	2,152,855	3,065,540	3,670,200	3,378,551	3,621,750	3,799,000	3,043,100	24,151,841
Per ton drawn..... do.	3.60	4.46	5.44	4.71	5.03	5.02	4.91	4.68
Timber used..... bd. ft.	450,898	614,739	700,858	510,953	497,690	365,740	210,089	3,629,196
Timber per ton drawn..... do.	0.754	0.89	1.04	0.713	0.691	0.484	0.34	0.70
Total man-shifts in mine.....	50,678	73,181	81,257	63,945	55,072	38,957	2/23,596	2/421,919
Ore per man-shift.....tons	11.8	9.4	8.4	11.2	13.08	19.41	3/26.23	3/12.22
Average no. men underground.....	171.7	220	237	186	163.1	117	71	163
No. of compensable accidents.....	34	56	74	59	41	27	11	332
Accidents per 1000 man-shifts.....	0.651	0.77	0.911	0.924	0.74	0.69	0.46	0.78
Powder ore break. diamond dr...lb.	-	-	-	-	-	-	108,650	108,650
Powder per ton broken.....do.	-	-	-	-	-	-	0.19	0.19

1/ Totals include figures for 1938.
2/ Exclusive of diamond-drill contractors.
3/ Development behind.

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TABLE 4. - Labor, power, and supplies per ton hoisted, Holden mine, January 1946 to August 1, 1946

Tons broken - 264,765 Tons from stopes - 342,885

	Development, MH/ton	Mining, MH/ton	Total, MH/ton
Labor (M H/ton)			
Breaking	0.0398	0.0053	0.0451
Drawing	-	.0615	.0615
Timbering	.0623	-	.0623
Mucking	.0599	.0057	.0656
Trans.	.0010	.0178	.0188
Haulage and Hoisting	.0084	.0466	.0550
Supervision	.0263	.0029	.0292
General	.0114	.0656	.0770
Total labor underground	.2091	.2054	.4145
Average tons per man-shift			19.3
Underground labor, percent of total cost			51.3
Man-shifts per day on surface, chargeable to underground operation			25.0
Power and supplies			
Explosives.....lb./ton	0.118	1.246	1.364
Timber, bd. ft. per ton	.860	.026	.886
Power..... kw.-hr./ton			
Air comp.	3.08	3.25	6.33
Hoisting	.20	1.12	1.32
Pumping	.01	.26	.27
Ventilation	.08	2.23	2.31
Lighting	.01	.15	.16
Slushing	-	.52	.52
Miscellaneous	.03	.81	.84
Total kw.-hr./ton	3.41	8.34	11.75
Contract diamond drilling and loading, percent		12.6	12.6
Percentage of other supplies to total supplies			23.4
Supplies and power - percent of total cost			36.1

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TABLE 5. - Development footage per ton

	1939	1940	1941	1942	1943	1944	1945
Tonnage drawn.....	597,787	687,729	685,903	715,806	720,240	756,331	618,927
Current dev. footage...	6,650	11,068	12,561	10,008	8,921	3,766	2,566
feet per ton drawn...	0.012	0.017	0.019	0.014	0.013	0.005	0.004
Stope dev. footage.....	6,654	12,001	12,923	9,165	12,452	12,363	2,674
feet per ton drawn...	0.012	0.018	0.019	0.013	0.017	0.017	0.004
Exploration footage....	5,312	6,725	5,123	1,232	-	-	-
feet per ton drawn...	0.010	0.010	0.008	0.002	-	-	-
Cu.yd., in linear ft...	-	-	6,394	-	2,709	2,387	851
Total footage.....	18,616	29,872	37,033	20,405	24,082	18,617	6,091
Total ft. per ton drawn	0.034	0.045	0.054	0.0285	0.033	0.025	0.010
	Development per man-shift						
Man-shifts dev. crew...	-	-	33,215	21,931	24,650	17,034	6,466
Footage per man-shift	-	-	1.11	0.93	0.98	1.09	0.94
Prod. and gen. crew - man-shifts.....	-	-	48,312	26,269	34,780	27,054	22,954
Total underground crew - man-shifts.....	-	-	81,527	48,200	59,430	44,088	29,420

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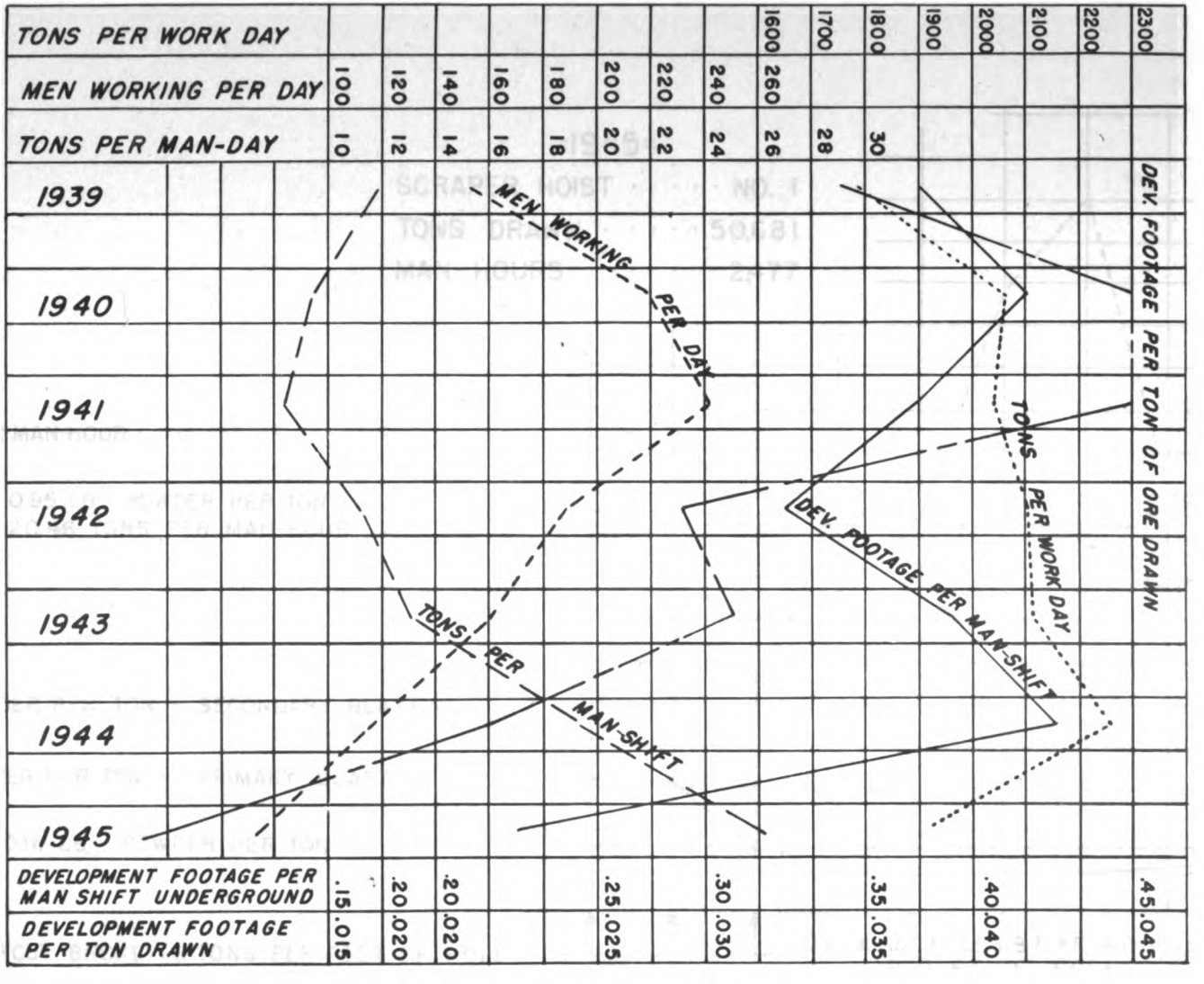
TONS PER WORK DAY
MEN WORKING PER DAY
TONS PER MAN-DAY
100 120 140 160 180 200 220 240 1600 260 1700 28 1800 30 1900 2000 2100 2200 2300
10 12 14 16 18 20 22 24 26
1939
1940
1941
1942
1943
1944
1945
DEV. FOOTAGE PER TON OF ORE DRAWN
MEN WORKING PER DAY
TONS PER WORK DAY
DEV. FOOTAGE PER MAN-SHIFT
TONS PER MAN-SHIFT
DEVELOPMENT FOOTAGE PER MAN SHIFT UNDERGROUND
.15 .20 .20 .25 .30 .35 .40 .45
DEVELOPMENT FOOTAGE PER TON DRAWN
.015 .020 .020 .025 .030 .035 .040 .045

Figure 18. - Operation efficiency.

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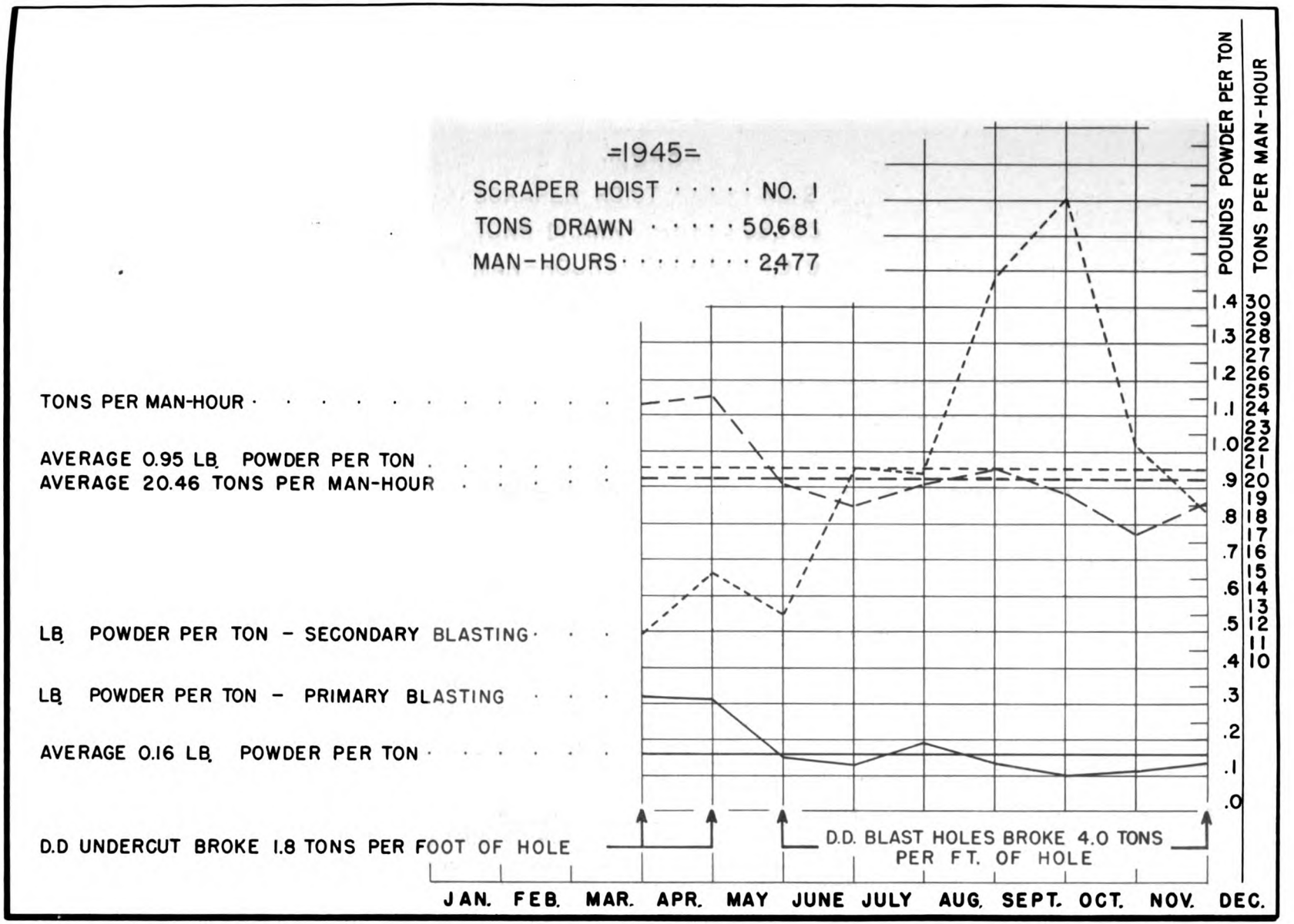
=1945=
SCRAPER HOIST ..... NO. 1
TONS DRAWN 50,681
MAN-HOURS 2,477
TONS PER MAN-HOUR
AVERAGE 0.95 LB. POWDER PER TON
AVERAGE 20.46 TONS PER MAN-HOUR
LB. POWDER PER TON - SECONDARY BLASTING
LB. POWDER PER TON - PRIMARY BLASTING
AVERAGE 0.16 LB. POWDER PER TON
D.D UNDERCUT BROKE 1.8 TONS PER FOOT OF HOLE
D.D. BLAST HOLES BROKE 4.0 TONS PER FT. OF HOLE
POUNDS POWDER PER TON
1.4 1.3 1.2 1.1 1.0 .9 .8 .7 .6 .5 .4 .3 .2 .1 .0
TONS PER MAN-HOUR
30 29 28 27 26 25 24 23 22 21 20 19 18 17 16 15 14 13 12 11 10
JAN. FEB. MAR. APR. MAY JUNE JULY AUG. SEPT. OCT. NOV. DEC.

Figure 19. - Hoist 1; record of man-hours and powder used per ton of ore drawn, stope 1775-247.

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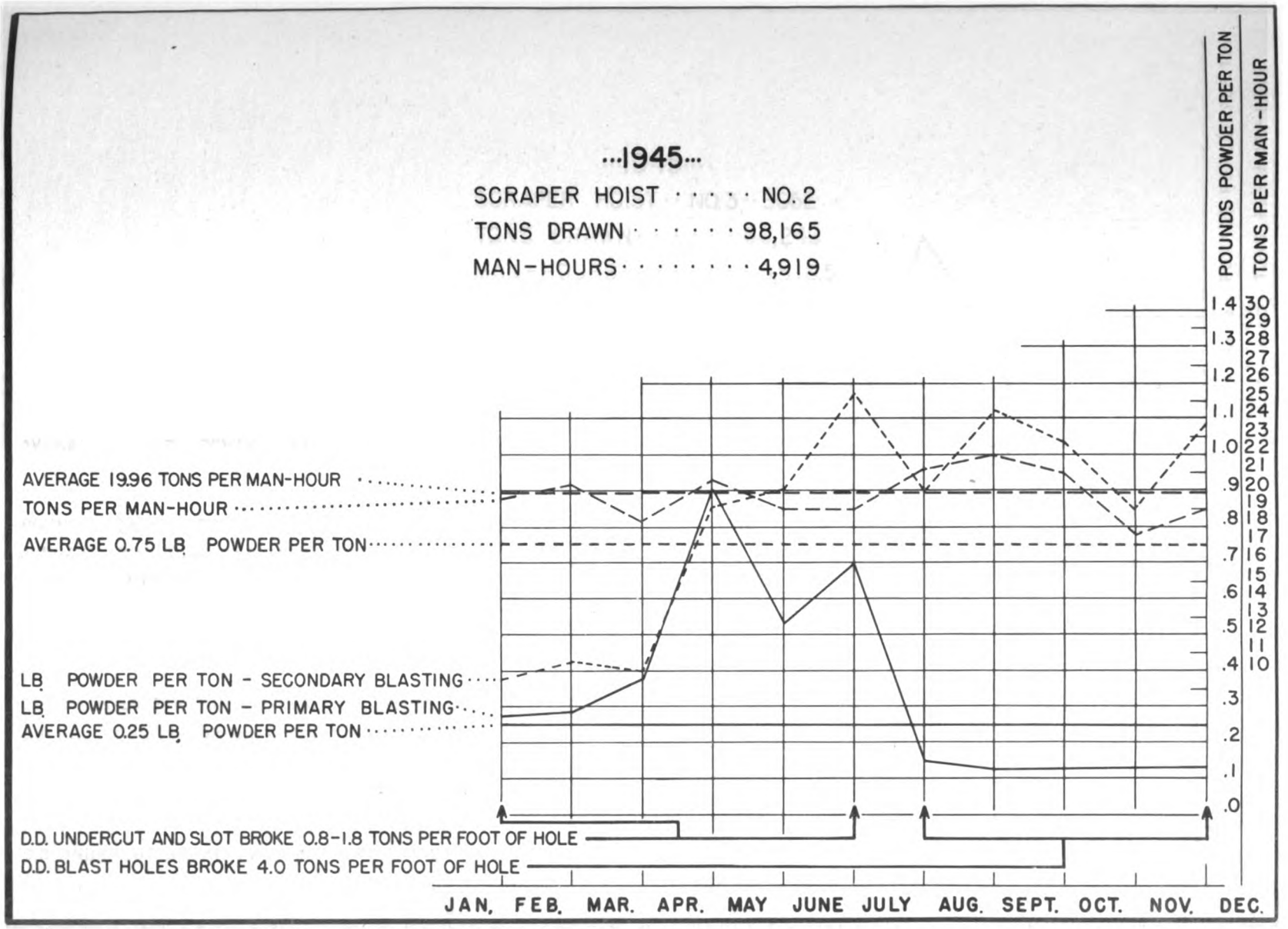
...1945...
SCRAPER HOIST ..... NO. 2
TONS DRAWN 98,165
MAN-HOURS 4,919
POUNDS POWDER PER TON
TONS PER MAN-HOUR
1.4 1.3 1.2 1.1 1.0 .9 .8 .7 .6 .5 .4 .3 .2 .1 .0
30 29 28 27 26 25 24 23 22 21 20 19 18 17 16 15 14 13 12 11 10
AVERAGE 19.96 TONS PER MAN-HOUR
TONS PER MAN-HOUR
AVERAGE 0.75 LB. POWDER PER TON
LB. POWDER PER TON - SECONDARY BLASTING
LB. POWDER PER TON - PRIMARY BLASTING
AVERAGE 0.25 LB. POWDER PER TON
D.D. UNDERCUT AND SLOT BROKE 0.8-1.8 TONS PER FOOT OF HOLE
D.D. BLAST HOLES BROKE 4.0 TONS PER FOOT OF HOLE
JAN. FEB. MAR. APR. MAY JUNE JULY AUG. SEPT. OCT. NOV. DEC.

Figure 20. - Hoist 2; record of man-hours and powder used per ton of ore drawn, stope 1775-257.

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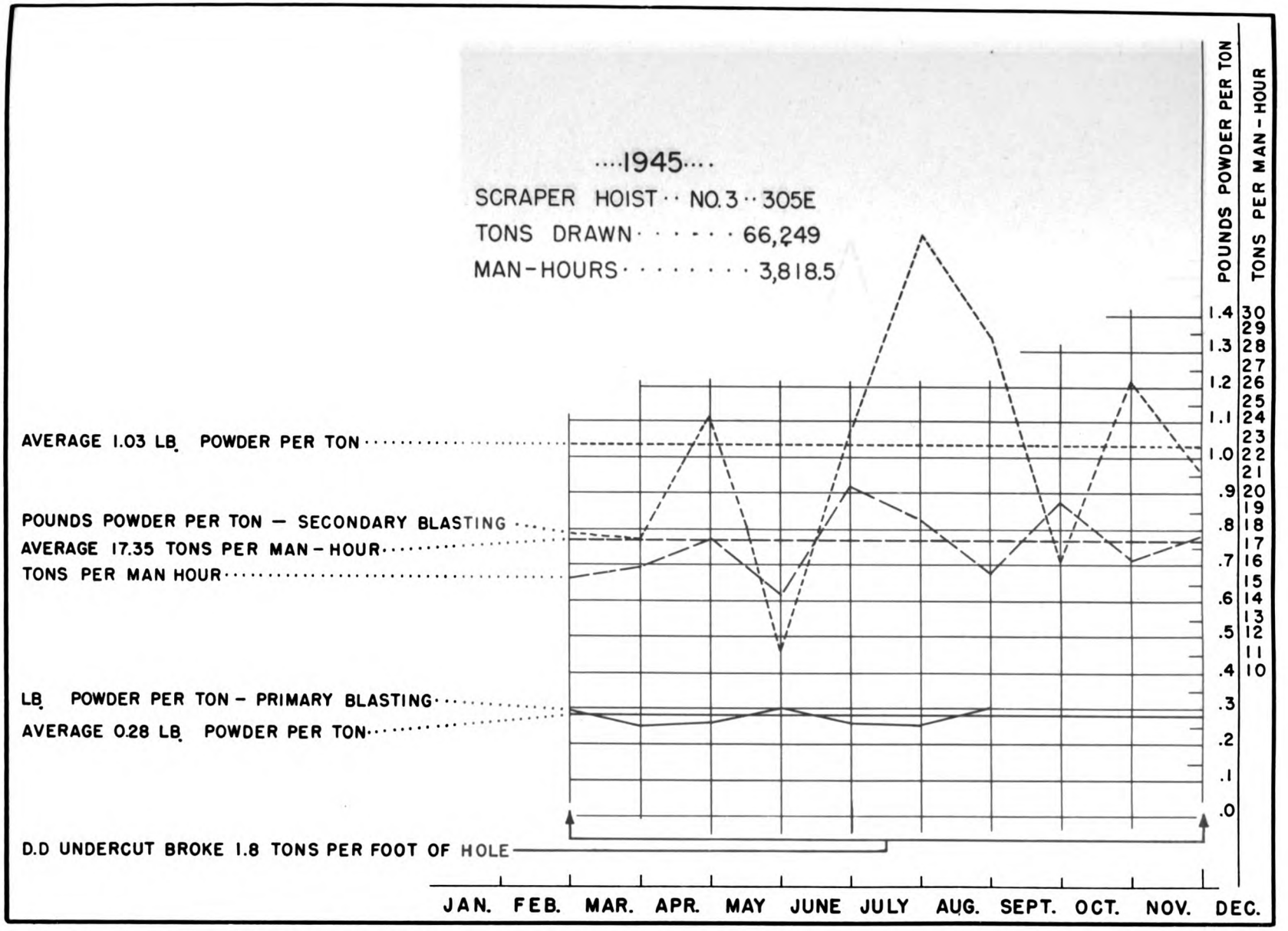

Figure 21. - Hoist 3; record of man-hours and powder used per ton of ore drawn, stope 1775-310.

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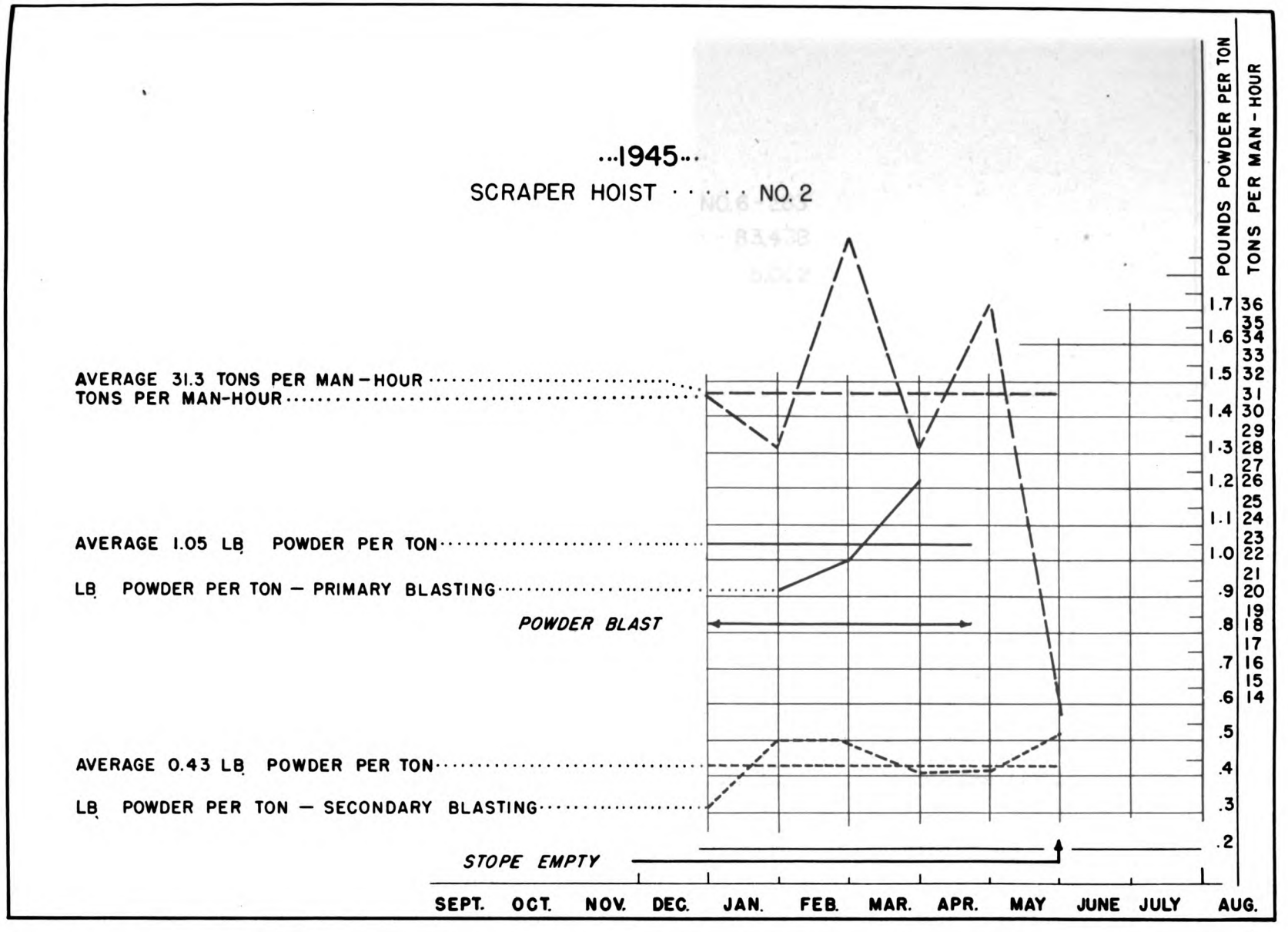
...1945...
SCRAPER HOIST ..... NO. 2
AVERAGE 31.3 TONS PER MAN-HOUR
TONS PER MAN-HOUR
AVERAGE 1.05 LB POWDER PER TON
LB POWDER PER TON — PRIMARY BLASTING
POWDER BLAST
AVERAGE 0.43 LB POWDER PER TON
LB POWDER PER TON — SECONDARY BLASTING
STOPE EMPTY
SEPT. OCT. NOV. DEC. JAN. FEB. MAR. APR. MAY JUNE JULY AUG.
POUNDS POWDER PER TON
1.7 1.6 1.5 1.4 1.3 1.2 1.1 1.0 .9 .8 .7 .6 .5 .4 .3 .2
TONS PER MAN-HOUR
36 35 34 33 32 31 30 29 28 27 26 25 24 23 22 21 20 19 18 17 16 15 14

Figure 22. - Hoist 2; record of man-hours and powder used per ton of ore drawn, block I, 1100-foot level.

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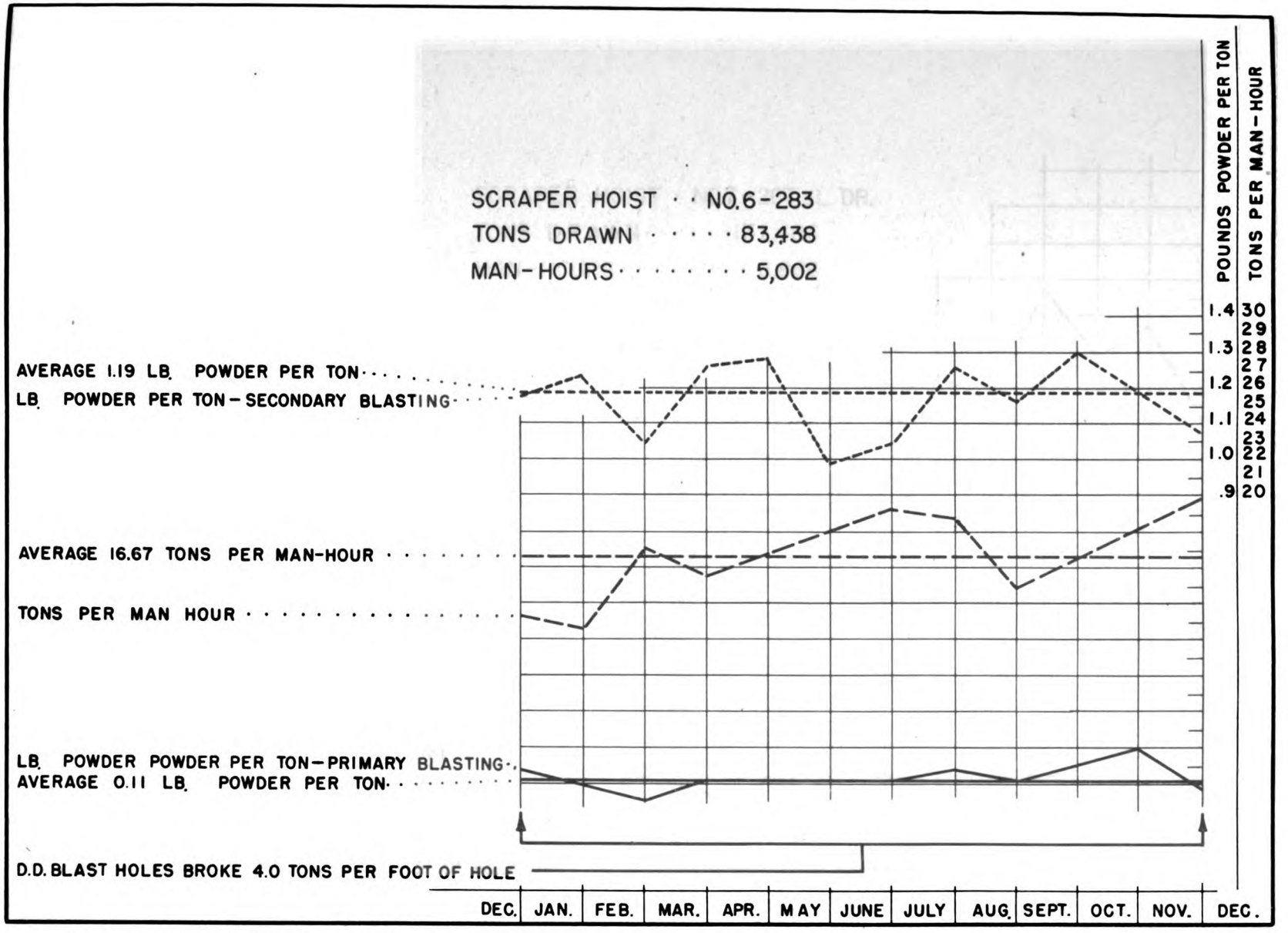
SCRAPER HOIST · · NO.6-283
TONS DRAWN · · · · · 83,438
MAN-HOURS · · · · · · · · 5,002
AVERAGE 1.19 LB. POWDER PER TON · · · · · · · ·

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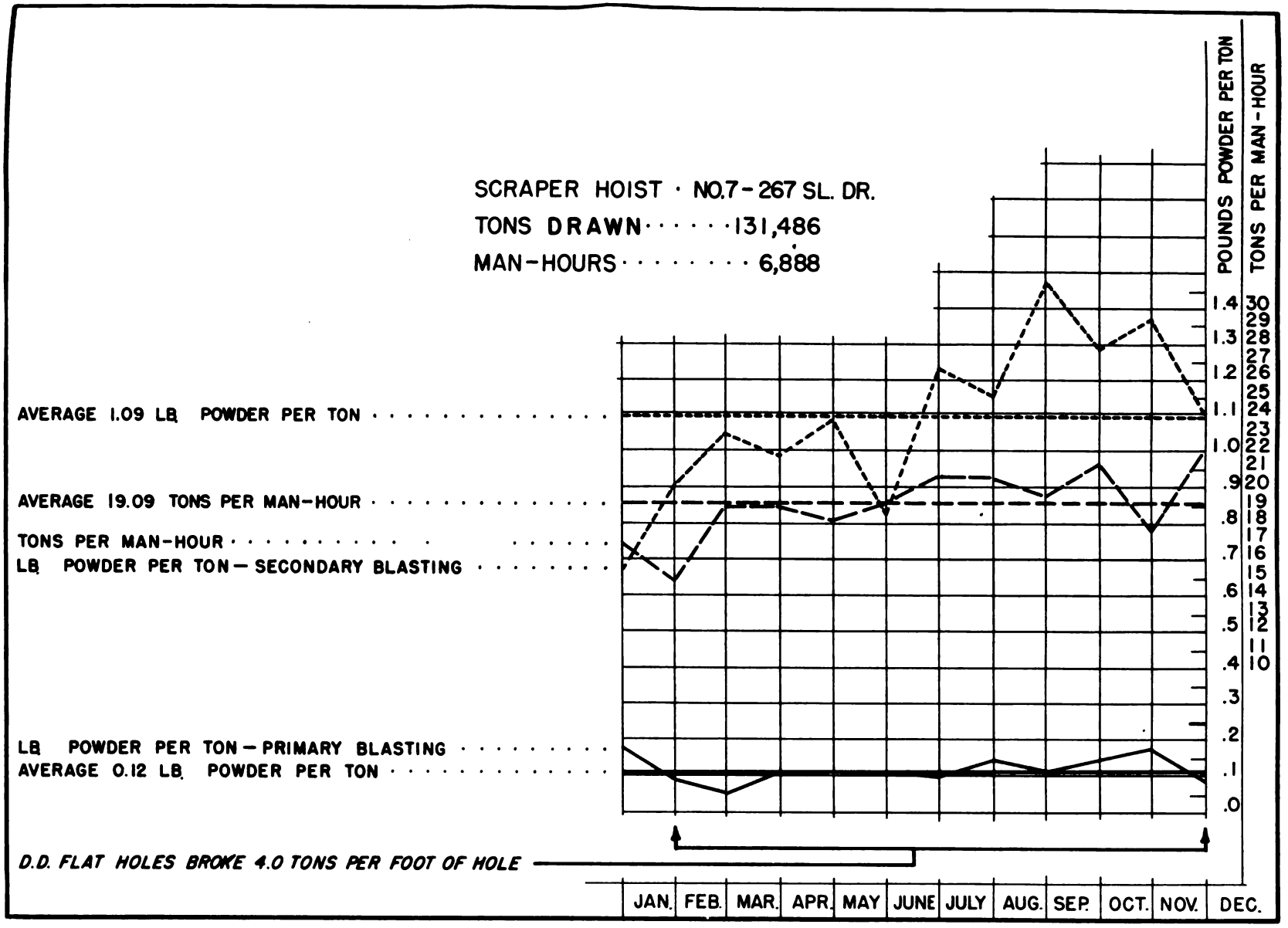
SCRAPER HOIST · NO.7-267 SL. DR.
TONS DRAWN · · · · · 131,486
MAN-HOURS · · · · · · · · 6,888
AVERAGE 1.09 LB. POWDER PER TON · · · · · · · · · · · · · · ·
AVERAGE 19.09 TONS PER MAN-HOUR · · · · · · · · · · · · · · ·
TONS PER MAN-HOUR ·
LB. POWDER PER TON — SECONDARY BLASTING · · · · · · · · · · ·
LB. POWDER PER TON — PRIMARY BLASTING · · · · · · · · · · · ·
AVERAGE 0.12 LB. POWDER PER TON · · · · · · · · · · · · · · ·
D.D. FLAT HOLES BROKE 4.0 TONS PER FOOT OF HOLE
POUNDS POWDER PER TON
1.4 1.3 1.2 1.1 1.0 .9 .8 .7 .6 .5 .4 .3 .2 .1 .0
TONS PER MAN-HOUR
30 29 28 27 26 25 24 23 22 21 20 19 18 17 16 15 14 13 12 11 10
JAN. FEB. MAR. APR. MAY JUNE JULY AUG. SEP. OCT. NOV. DEC.

Figure 24. - Hoist 7; record of man-hours and powder used per ton of ore drawn, stope 1775-280.

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